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Research Clubs at Mills.

An opportunity for experiment and investigation exists at most metallurgical plants of any magnitude, which requires only a little enthusiasm and direction on the part of a leader to develop results that will be well worth the effort. We do not refer to the elaborate testing departments and research laboratories maintained by the larger companies, but more especially to less formal associations in which the esprit de corps is the basis of organization and provides the needed incentive for their work.

Conditions are favorable for such an organization at almost every plant of importance. In the first place, it may be assumed that every man employed is, or should be, interested not only in the routine work but in his own progress and education as well. As most mills are operated twenty-four hours in the day, it is possible to carry on some experimental processes continuously by having men on each shift give a little time and attention thereto. And last, but not of least importance, such an organization would be of special value to men working at mills which are more or less isolated from other centers of interest. The social feature of life at these places is generally recognized by companies, and clubs usually are provided for the men. It would not detract seriously from the relaxation which the club is designed to afford, if say, one evening a week were devoted to an informal discussion of professional work and if this interchange of ideas was further extended to planning experiment and research.

Both the men and the company would profit from such a plan. To the former it would be a stimulus, creating genuine personal interest in the work and bringing out latent ability. This would enlarge their view and offer opportunity for advancement. The benefit to the company would lie in improved work, cheapened cost of operation, elimination of unnecessary steps in the process, and a spirit of co-operation among its employees. Even more substantial results might follow, such as the successful treatment of an ore formerly refractory to the process employed.

As said before, a leader who can suggest lines of investigation is invaluable in such a plan, and the company must acquiesce in making the necessary expenditures. We recall several instances where this arrangement is working successfully. At Goldfield the Florence mill is equipped with a special laboratory where the men are encouraged to make experiments bearing on the cyanidation of gold ores. The Portland Metallurgical Society was formed of employees of the Portland Gold Mining Company, and was the direct means of evolving the process now used by the company at its Victor mill. Recently Mr. Dorr has organized his men at the Mogul mill in South Dakota into a research club from which good results are expected. Doubtless other instances of clubs organized on this plan could be found, but there is still room for many more.

Continuous Cyanidation.

Hitherto in these columns emphasis has been laid on the value of continuity in metallurgical processes. Compared with intermittent procedure, the continuous method gives greater service per unit of time and labor, adds to net profits and reduces the losses sustained in the repetition of an intermittent method. In two of the principal steps in the cyanide process, agitation and filtration, we have seen remarkable progress owing to the introduction of the continuous feature, and this idea has persisted in the minds of metallurgists so that they are endeavoring to secure continuous operation wherever possible.

In this connection we are glad to present to our readers a timely article on continuous cyanidation, by Mr. John E. Rothwell, who has been studying and developing this idea for several years. A study of the flow sheet accompanying the article will reveal the advantages claimed by the author for the method. Mathematical accuracy and definiteness are substituted for guesswork and rule-of-thumb procedure. The operator may know just what is going on in any part of the process at any time. By means of measuring devices, such as diaphragm pumps, the quantities of solution or pulp delivered from one point to another are determined exactly. There is a definite quantity of solution of definite value drawn from the system for precipitation of the dissolved values. And yet, with all the mathematical nicety which marks continuous operation, there is ample elasticity to provide for accidents or other contingencies in that part of the treatment which concerns extraction by cyanide.

The complete absence of filters in this method is a notable feature considered from the financial standpoint. The cost of installation of a continuous treatment method probably would be less, and certainly not more, than that of a plant using filters, while the cost of subsequent operation certainly would be less. Extraction also should be improved owing to the manner of applying the countercurrent of solution to the ore. It will be noticed that the weakest and lowest grade solution is applied to the ore when it has been almost exhausted of its precious metal content, and that the solutions can be strengthened by the addition of cyanide as they come in contact with richer ore. As pointed out by the author, this feature insures the highest efficiency from the solvent.

As a contribution to progress in cyanidation the idea of continuous treatment is very timely, and we invite metallurgists to a free discussion of its merits.

The Cyanamide Process in Gold Metallurgy.

Experimental work with the Clancy cyanamide process continues to absorb the attention of metallurgists in the West. The experiences are varied, and the indications are that the ordinarily capable experimenter is not equipped, either by education or experience, to account for all the results he obtains, or to obtain uniform results continuously. This does not reflect unfavorably on the merits of the Clancy process, nor does it derogate from the correctness of the underlying principles. It mainly calls attention to the fact that the chemistry of the organic compounds of gold is not as well known as it will be a few years hence, and that the conditions for successful operation may be confined within comparatively narrow limits.

It is, therefore, timely to caution against promiscuous experimentation, as well as against the error of drawing hasty conclusions from a few results. There are so many variables to be taken into consideration that it requires patient, continuous and intelligent experimenting to determine them singly and in combination. The difficulties are increased because the present accumulation of knowledge on the subject is small, and hence each investigator must work according to the best light he can obtain. It would be of great benefit to the fraternity if metallurgists would contribute their results; not putting them forward as from ripe experience, but as tentative statements of observation.

One of the difficulties encountered is the satisfactory chemical control of the process. We are informed that the titration of regenerated cyanide solutions offers some difficulty. Results are varied, and for reasons not always apparent. It is stated that some regenerated cyanide solutions will show a decided solvent action on gold and silver, but that the same solutions will not give a titration for cyanide according to the usual methods. What condition exists in these solutions? What is the solvent, and how is it to be determined as regards quality and quantity? What are the effects of various current densities, and what arrangement and size of electrodes will give the best results? Can regeneration of the cyanide solution be well accomplished in contact with the ore pulp, or is it better effected in clear solution after filtration?

These and many other questions have confronted the present investigators, and the answers are not yet made to their satisfaction. The free exchange of opinons and results will solve these points, just as similar problems were solved in the standard cyanide process which has been brought to such a high state of perfection.

Co-operation and Competition in the Iron and Steel Industry

The public was given further enlightenment last month as to the attitude of Mr. E. H. Gary, chairman of the United States Steel Corporation, with respect to the corporation's position and the campaign of price maintenance, technically known as "co-operation," which he has conducted so successfully for several years. Inasmuch as Mr. Gary's attitude has not changed of late, and as the pursuance of the campaign necessitated his expounding his views from time to time, there was nothing strictly for him to say on these subjects when he appeared before the Stanley committee of the House of Representatives, pursuing an investigation of the affairs of the United States Steel Corporation and its relations with the independent manufacturers. However, the occasion was afforded for Mr. Gary to re-state his views, fully and distinctly, and he made most of his opportunities.

The policy of "co-operation" is offered as a modern invention, a substitute for the old practice of "destructive competition." The work is conducted very simply, the steel manufacturers merely meeting from time to time and then and upon other occasions freely exchanging information as to their business affairs, there being in particular an arrangement

whereby, while no manufacturer is bound not to reduce his prices, he is expected to communicate news of his intention to the other manufacturers before he announces the reduced price to purchasers. The policy is simplicity itself, being essentially based on a frank and full exchange of information between the different manufacturers in the iron and steel industry.

Mr. Gary was candid, apparently, to the members of the committee, startlingly candid, when he expressed his strong wish that there should be a department of the government to which the management of the Steel Corporation could go when in doubt as to what the law allowed it to do, and, upon giving adequate information as to its position, receive counsel which if followed would relieve it from responsibility. We opine that Mr. Gary's attitude in this matter is due to his having stronger views as to the scope and bearing of the Sherman anti-trust law than are entertained in many quarters. There is no reason to infer that Mr. Gary courts government supervision, with regulation of prices, for any other reason than because of the potency of the Sherman law or the force of public opinion, which would be likely to make another law were it not in existence.

To draw a word picture illustrating the differences between the so-called "destructive competition" and this modern policy of "co-operation" may readily appeal to the senses, but it is not stating or developing a scientific law. What is this "destructive competition"? Is it not destroying that which is inefficient and making more durable that which is efficient? This in itself is more likely to be good than bad. It is true Mr. Gary suggested that without the formation of the Steel Corporation there was danger of a monopoly being created, for he stated as his firm belief that had the Carnegie Steel Company continued in its old position and with its old management it would have driven all other steel producers out of the field. We can hardly concur in that belief. Prior to the formation of the Steel Corporation, April 1, 1901, the Carnegie Steel Company was making 2,500,000 tons of steel, which at that time represented approximately one-fourth of the country's steel ingot production of 10,000,000 tons. The country's production last year was in excess of 25,000,000 tons. Certainly a war of destructive competition would have tended to increase, rather than decrease, the demand, and it is inconceivable that a single concern could carry on such a warfare and at the same time accumulate funds sufficient to multiply its productive capacity by ten.

Mr. Gary's testimony brought out what was emphasized at the time the Steel Corporation was formed that one of its objects was to forestall a large amount of new erection which was planned by the various interests consolidated, new erection far beyond what seemed to be in keeping with the country's requirements and which would probably have involved the most serious competition. Quite likely it was an economic gain to the country that such unnecessary duplication of plants should be avoided. In the light of the actual history of the iron and steel industries during the last years this economic gain, as a consequence of the formation of the Steel Corporation, can hardly be denied.

Even granting that the "destructive competition" is altogether bad, the history of the past few years does not necessarily indicate that the new principle which it is sought to put in its place is necessarily good. We observe in the first place that there was a period of three years, from late in 1904 until late in 1907, when demand was sustained and the increasing productive capacity of the iron and steel industry was continuously and fully engaged. During that period it was urged that steady prices meant steady demand, but in the retrospect it appears that the slightly advancing tendency which the market displayed during that period tended to encourage buying and in some cases induced the anticipation of wants. Usually there are some who wait, but these did not and the result was a twelvemonth in which demand was scarcely more than half what it had been in the preceding twelvemonth. Never before had there been so violent a decrease in demand and this despite the ever repeated preachment that steady prices meant steady demand.

A still more serious feature, however, is this, that the encouragement of buying in 1906 and 1907 created a false impression as to the ordinary iron requirements of the country; it set a standard which the trade expected to be lived up to nearly all the time. There was an inducement to new erection, and the policy of co-operation undoubtedly added a great incentive, for it involved, directly or indirectly, the maintenance of prices and the distribution of whatever demand there was along the different producers according to their capacity. Naturally the larger one's capacity the larger the share of the going business. Accordingly we find the industry's capacity, including a few furnaces not yet completed, at about 35,000,000 tons of pig iron per annum, practically 50 per cent in excess of the average demand in the three years of extreme prosperity, 1905-6-7, and still more in excess of the demand which has been experienced more recently in the past six or nine months.

Thus we cannot avoid the conclusion that if the policy of "co-operation" has eliminated the old fashioned "destructive competition" it has instituted a practice of unnecessary new erection. We are not prepared to give unqualified approval to either result.

It is said that the country must "grow up" to the new capacity. In point of tonnage perhaps it will. Certainly it will if the old law of increase in tonnage demand holds good, but we are not certain that it will. The requirements, in service to be performed, presumably will, but nothing is more certain than that much of the steel now being employed is not as suitable as might be for the purposes to which it is applied. Frequently the service can be more effectively performed by lighter but better steel, and we may find that, while the service performed by the new steel made doubles in the future as often as in the past, the tonnage required to perform the increased service will not increase in proportion. If that is so much of the new construction now indulged in, anticipating future requirements, is misplaced, whereas by waiting until the demand appears the investment could be made much more intelligently and with much greater effectiveness.

And yet, after all that may be said on the shortcomings of the policy of co-operation, the fact remains that this policy with its fixation of prices is in line with the tendency of the times toward scientific socialism. And as such it is not only logical but eminently practical.

Indianapolis Meeting of the American Chemical Society.

The Indiana Section of the American Chemical Society is making special efforts to make the summer convention unusually successful. This will begin on Tuesday, June 27, and last throughout the remainder of the week. The council will meet on the afternoon of Tuesday, June 27, and after a business meeting will be taken for an automobile ride to the Indianapolis Speedway.

On Wednesday the general session will begin and three general addresses will be given. Prof. Charles Baskerville, of the College of the City of New York, will speak on "The Chemistry of Anaesthetics." Prof. Edward Kremers, of the University of Wisconsin, will follow with a paper on "The Quinhydrone Hypothesis of Plant Pigmentation." The third address, by Prof. W. Lash Miller, of the University of Toronto, will be upon "The Chemical Philosophy of the High-School Textbooks."

These papers will be followed by a complimentary luncheon at the German House, and after luncheon meetings of the Agricultural and Food, Industrial, Organic, and Physical and Inorganic Divisions and the Biological Section will be held. In the evening a complimentary smoker will be held in the gardens of the German House.

On Thursday morning there will be meetings of the Agricultural and Food, Industrial, Organic, Pharmaceutical, and Physical and Inorganic Divisions, one of the special features being the meeting of the Pharmaceutical Division in the new Science Building of the Eli Lilly & Company, followed by a luncheon there to the pharmaceutical chemists. At the same time a complimentary luncheon will be given to the remaining chemists at the Indianapolis Brewing Company, followed by an inspection of their plant. In the afternoon all of the chemists will visit the plant of the Eli Lilly & Company.

On Thursday evening there will be a public lecture by Mr. A. D. Little at the German House on the earning power of chemistry. This lecture is one of the features of the meeting and is intended to interest chemists, business men and the general public alike. Following the lecture there will be a concert in the gardens of the German House.

On Friday there will be an early visit to the Indianapolis Water Works followed by meetings of the Fertilizer, Industrial, Physical and Inorganic Divisions and the Sections of Biological and of India Rubber Chemistry. At 12:30 these will be followed by a complimentary luncheon at the automobile factory of Nordyke & Marmon, followed by an inspection of their factory. Later in the afternoon visits will be made to the Encaustic Tile Company, to the Fairmount Glass Works and to E. C. Atkins & Company. In the evening the usual banquet of the summer meeting will be held.

On Saturday there will be visits to the Kingan & Company, Ltd., to the Polk Sanitary Milk Company and to the Diamond Chain Company; in the afternoon to the Van Camp Packing Company.

A special endeavor is also being made to see that ladies accompanying members have an enjoyable summer outing including automobile rides, receptions, theater parties and luncheons.

Over 150 papers have already been received for the divisional meetings.

Dr. Charles L. Parsons, Durham, N. H., is the secretary of the Society.

The Western Metallurgical Field.

The Miami Mill.

The Miami copper mill in Arizona is one of the large plants that recently entered the list of producers. The general line of ore treatment is not dissimilar to that adopted in the other large copper mills treating disseminated, low-grade copper ores. The following outline was given at the May meeting of the New York Section of the Mining and Metallurgical Society of America by Mr. J. Parke Channing:

The ore is crushed in gyratory crushers to 2-in. cubes and screened through cast-steel trommels having ¾-in. holes. The oversize is reduced to ½-in. cubes by 54-in. x 24-in. rolls, and is added to the undersize of the trommel. This is elevated and distributed to six 1000-ton bins, each representing a unit in the mill. The ore is automatically weighed and sampled as it leaves these bins, and then flows into a shaking launder with screen bottom, where water is added. The oversize, larger than ⅛ in., is then reduced in Burch high-speed rolls and added to the launder undersize. The mixture is then divided, one-half going to each of the two sections of the unit.

The treatment in a section follows: Over two Callow screens of a mesh of 0.024 in., oversize to a 6-ft. Chile mill, at present provided with 0.029-in. screen. The two products join and go to a drag classifier, which consists of an endless rubber belt with blades on the bottom. The coarse material from the drag classifier goes to a 10-spigot Richards hindered settling hydraulic classifier, the discharge from each spigot going to a Deister No. 2 sand table. Each table makes concentrate middling and tailing products. The middlings and certain tailings are ground in a 6-ft. Hardinge mill and classified into sand and slime, the sand being treated on two Deister sand tables, and the slime on a Deister slime table.

The overflow from the drag classifier joins the overflow from the end of the Richards classifier and goes to a large conical dewatering tank. The underflow from this is treated on Deister slime tables which make concentrates, middlings and tailings. The middlings are retreated on Deister tables. The overflow from the large settling tank flows to eight smaller conical tanks, the underflow of which is treated on Deister tables, making concentrates, middlings and tailings. The middlings join the coarser slime middlings and are treated on the two Deister slimers mentioned above.

The water recovery is accomplished as follows: The mill tailing goes to large wooden conical tanks, underflow to waste launder and overflow to 80 concrete settling tanks. Overflow water returns to the mill, underflow joins the sand tailings and goes to tailings pond. This pond is provided with a dam made of tailings, back of which the water containing slime is settled and pumped to the mill.

The mill feed has averaged about 2.6 per cent copper, which is higher by nearly 1 per cent than the ore treated by some of the other large copper mills, notably the Utah Copper Company's mills. Ratio of concentration is 20:1, and the extraction has been gradually climbing from 70 per cent to 78 per cent, and it is expected that the promised 80 per cent will be reached. The concentrate runs about 40 per cent copper, the mineral being nearly pure chalcocite with but a small amount of pyrite.

The mill consists of six sections, of which three are equipped with two 6-ft. Chile mills each. The fourth section is provided with three Burch high-speed fine rolls, and they will shortly be used for fine crushing for comparison with Chile mills. The Burch roll is provided with a fleeting movement by which one roll is moved laterally in front of the other in the direction of the axis of the roll so as to prevent grooving.

The mill was originally designed so that the six sections would have a capacity of 2000 tons per day, but it seems evident that this will reach 3000 tons per day. The power consumption is estimated by Mr. Channing to be about 1000 kw, or 1400 hp, for the treatment of 2000 tons per day in four sections. This will include the crushing and pumping, each of these operations being done for about 8 or 10 hours alternately. The mill itself will take about 720 kw, leaving 280 kw for crushing one-half of the day and for pumping the other half, so as to keep the load steadily at about 1000 kw.

Chlorination Abandoned.

It is generally understood that the chlorination plant of the Portland Gold Mining Company at Colorado Springs will be

remodeled for use as a cyanide plant. The company treats Cripple Creek ore, and is using a combination of the chlorination and cyanide processes. The raw ore is first ground and roasted, then treated by chlorination, the tailings being concentrated on tables. The table concentrates are shipped to smelters, and the table tailings classified into sand and slime. The sand is ground in tube mills in cyanide solution, and the gold solutions recovered by decantation and precipitated on zinc shavings. In addition to treating the chlorination tailings by cyanidation, the company has treated a considerable tonnage of its old accumulated tailings by the same method. The total tonnage daily treated by cyanide has been about 500 tons, equally divided between current and old tailings.

The decision to abandon the chlorination end of the treatment at the Portland will leave but one company still treating Cripple Creek ore by chlorination, viz., the United States Refining & Reduction Company. At this company's Standard Plant at Colorado Springs a combination of chlorination and cyanidation is used, on much the same order as at the Portland, with the exception that at the Standard chlorine is generated electrolytically instead of by the action of sulphuric acid on bleaching powder. Cyanide metallurgists may glean a bit of satisfaction from the announcement that chlorination is to be abandoned by one of the old companies, but it must be remembered that chlorination has had a long and honorable record in connection with Cripple Creek ores, and that cyanidation has only in recent years arrived at that state of perfection which would make it a favored rival. True, some have contended almost from the beginning that cyanide was a better agent than chlorine for treating these ores, but in the past there has been room for argument which cannot exist at the present time.

Colorado.

The complaint against the backward state of mining in Colorado, which has been heard for the last year or two, is gradually giving place to a more hopeful expression. It would be difficult to state concisely the reasons for the decline, but undoubtedly the wonderful interest in agriculture and irrigation, the restrictions against prospecting on the public domain and the high cost of equipping and handling low-grade properties have all had their effect. Now, however, the pendulum seems to be swinging back to mining, and there is a noticeable increase in interest. The commercial bodies are talking mines and mining, and public excursions are being conducted into the mining districts. Irrigation has had its sway and mining is again in a fair way to receive respectful attention from financial sources.

By way of publicly encouraging the prospector to open new fields the Denver Chamber of Commerce has raised a fund which is available to prospectors who wish to go out on a "grub stake" agreement. The idea is that a prospector will be outfitted and kept in the field during the season, and that any discoveries shall become the joint property of the Chamber and the prospector. The Chamber retains the major interest, and has the work in charge of a committee of its members. This committee consists of Messrs. James Doyle, Max Boehmer and Ernest L. Foster, all capable mining men and engineers who have had a long and active experience in mining in the State. Opinion is divided as to the practicability of the plan, but it is a worthy effort to revive interest in an industry which, though not comparable in its magnitude to that of 10 or 20 years ago, still holds a high rank in the principal industries of the State.

The Colorado School of Mines' thirty-seventh annual commencement occurred at Golden on May 26. Thirty-five young men were graduated and received degrees in mining and metallurgy. Among the theses presented as part requirement for graduation, two received prizes which are awarded annually. Mr. Roy F. Smith received the Brunton transit, and Messrs. E. J. Dittus and R. G. Bowman received the Stoiber prize of $150 for the best thesis in metallurgy. Their work was on the direct production of molybdenum steel in the electric furnace.

At the annual election of the faculty of the school ,there were some changes, though not as many as might have been expected in view of the persistent rumors to the effect that sweeping changes would be made. Prof. A. J. Hoskin and Assistant Prof. A. C. Smith, of the department of mining, and Assistant Prof. C. E. Smith, of the department of geology, resigned. Their places have not been filled.

The testing plant being erected at the school is under cover and will be completed before fall. The Legislature made an appropriation of $50,000 for this work and no delay will ensue if the sum proves sufficient.

The Iron and Steel Market.

The half year just ended has been an unsatisfactory one for the iron trade. The production of pig iron may be estimated at 11,750,000 tons, with a probable error of less than 250,000 tons. The three preceding half years showed heavier production, while the second half of 1905 and both halves of 1906 and 1907 showed heavier production. In other words, since the middle of 1905 production by half years has been heavier than in the half year just ended with the exception of 1908 and the first half of 1909.

This record is particularly unsatisfactory because there has been such a large increase in capacity in recent years. In the first half of 1905 there were made 11,163,175 tons of pig iron and in the second half 11,829,205 tons. These outputs represented substantially the total capacity available at the time, at any rate the capacity available with ordinary market prices, for prices were not unsatisfactory at the opening of that year and rose during the year. On the other hand, it was physically possible to make fully 5,000,000 tons more pig iron in the half year just closed than really was made, so that production was just 70 per cent of capacity.

Prices of open-market products were unsatisfactory in the half year. Pig iron, which at the opening of the year stood at a level affording only the most moderate profits to the most fortunate furnaces and represented an actual loss to a number of furnaces, particularly those in the Mahoning and Shenango valley, declined slightly, the average level being about 25 cents a ton lower on July 1 than on Jan. 1. Connellsville coke advanced and then declined, showing practically no net change, while scrap in most markets declined slightly.

The slight decline in the open-market commodities, considering that demand and production were so much below capacity, shows that the adjustment had already been made. Indeed, the year 1910 was one of steady and fairly rapid decline in prices in the open-market commodities.

In the finished steel trade there is a controlled market, and as a result a demand far below productive capacity has not produced a profitless market level. Apart from the profits involved in coke manufacture and ore production, the level has been such that there was a profit from purchased pig iron made into billets, and from purchased billets made into most finished steel products. Technically there were some advances in finished steel products early in the year, there being two of a dollar a ton each in wire products and one of 10 cents per box in tin plates. The wire advances, however, involved granting buyers large contracts at old prices, before the advances, while the advance in tin plates merely covered the advance in pig tin effected by the foreign syndicate, thus giving no benefit to the tin-plate mills. On the other hand, merchant steel bars dropped through the action of the Republic Iron & Steel Company on May 24 announcing a cut of $3 a ton in its price on steel bars, while as a result of a conference between all the important steel makers, except the Republic company, on May 29, reductions were made of $1 a ton in plates and shapes, $3

on blue-annealed sheets, $4 a ton on black and galvanized sheets and approximately $4 a ton on corrugated material. Technically, the average of all finished steel products dropped about a dollar a ton during the half year on account of these changes, but by reason of the peculiar conditions surrounding the wire and tin plate advances the effective decline was really more. Standard section rails are not included in this reckoning, as they stand in a class by themselves, the nominal price not having varied for ten years, while the quality has been materially improved.

Thus the finished steel trade suffered slightly reduced prices and materially reduced production, although its operations were still fairly profitable, and much more so than those of the merchant blast furnaces.

The trade encountered several severely adverse influences. On Feb. 22 the Interstate Commerce Commission handed down adverse decisions in the "advanced rate cases" when the general expectation had been that the railroads would be granted at least a material part of the advances in rates for which they had contended so long and earnestly, though perhaps not altogether wisely. These decisions cast a gloom upon the trade. The railroads at once curtailed purchases, and shortly afterwards a number of them suspended shipments on orders already placed.

The failure of the Sixty-first Congress to pass the Canadian reciprocity bill induced the President to call an extra session of the new Congress, which is Democratic in the House and not safely Republican in the Senate. The iron trade favors Canadian reciprocity, but became seriously disturbed over the prospect of tariff revision. May 15 the Supreme Court handed down its decision in the Standard Oil case. This was adverse to the "trust," but as the words "reasonable" and "undue" and the phrase "in the light of reason" were used in the decision there was a short-lived impression that a new and mild construction had been placed upon the Sherman law. The Tobacco decision a fortnight later, with more mature reflection upon the Standard Oil case, largely dispelled this impression, and the country became convinced that there may readily be found a number of business institutions in existence contrary to the Sherman law as interpreted.

The next adverse influence was the investigation of the United States Steel Corporation, intrusted to the Stanley committee by the House of Representatives.

In these adverse conditions the iron and steel trade has perhaps done fairly well in making the record it has in the half year. For the future, the second half of the year, the prospect is much brighter. No one can well maintain that the situation has not been quite thoroughly "liquidated." The buying trade has conducted itself with a deep impression of the adverse influences and is more likely to throw off the incubus than to yield further. It is the uniform experience that the influence of adverse developments lasts but a time, and the mental condition produced by a new condition wears away even though the condition itself may not be removed.

It is the general expectation in the trade that the second half of the year will show a much larger tonnage than the first half. There is no expectation of materially higher prices, although slight upturns in pig iron may occur in some districts. In finished steel the position is stable. While the Republic Iron & Steel Company withdrew formally from the councils of the steel manufacturers and took independent action in the matter of steel bar prices, it has not abandoned the general policy of co-operation. After the slight price cuts made in May most buyers adopted the view that other and deeper cuts would follow. This the steel makers insisted was not the case, and during the past month the expectation of buyers of lower prices has decreased. There has been a slight increase of buying of finished steel products, beginning about the second week in June, and with any further increase the stability of the finished steel price level may be regarded as reasonably assured.

Prices are quotable as follows:

Pig. iron, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; Bessemer, $15; basic, $13 to $13.25; No. 2 foundry, $13.50 to $13.75; malleable, $13.25 to $13.50; gray forge, $13 to $13.25.

Unfinished steel, Bessemer and open-hearth billets, $21; sheet bars, $22, both f.o.b. maker's mill, Pittsburgh or Youngstown; wire rods, $29, Pittsburgh.

Finished steel products, f.o.b. Pittsburgh, except as noted:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, 1.35 cents for tank quality, ¼ in. and heavier.

Shapes, 1.35 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles 2 in. x 3 in. and larger.

Steel bars, 1.25 cents, base; iron bars, 1.35 cents, Pittsburgh; 1.20 cents to 1.25 cents Chicago.

Wire nails, $1.80, base, per keg; plain wire, 1.60 cents, base; painted barb wire, 1.80 cents; galvanized barb wire, 2.10 cents.

Black sheets, 28 gage, 2 cents; galvanized, 3 cents; blue annealed, 10 gage, 1.50 cents; painted corrugated roofing, 28 gage, $1.40 per square; galvanized, $2.55.

Tin plates, 100-lb. cokes, $3.70 per base box.

CORRESPONDENCE.

The Manufacture of "Air-Salpeter" a Well Paying Industry.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The fixation of atmospheric nitrogen for fertilizer purposes is growing rapidly in importance, and it is interesting to notice that the production of nitrates from the air for fertilizer purposes is indeed a lucrative undertaking.

Its renumerativeness is plainly shown in the report presented recently at a stockholders' meeting by the president of the Norske Hydroelektriska Aktie Sellskap, Norway. This company was formed several years ago to develop and operate the Birkeland and Eyde nitrogen fixation process. This process, which is well known, consisted in an arc spread out into the shape of a thin disk by means of a magnetic field.

Recently this process has been abandoned, and the Schönherr process, of the Badische Anilin Fabrik has been installed and is exclusively operated at Notodden, Norway, where the company's factory is located.

The Norske Hydroelektriska Aktie Sellskap is capitalized at about $2,000,000, of which $1,200,000 is preferred and the remaining $800,000 is common stock. During last year dividends amounting to 8 per cent on the preferred stock and 5 per cent on the common stock were paid.

The gross earnings during the year amounted to $1,200,000, of which $350,000 was net income. Half of this sum was put aside for use in various funds, $23,000 was distributed among employees, and the remainder, after the dividend was paid, was transferred to the credit of 1911, to take care of the running expenses for this year. There are certainly few new chemical industries that can show any better financial record.

The company could not begin to satisfy the demand for calcium nitrate, or "Norge Salpeter," as its product is known to the trade, and on this account they are developing, in conjunction with other financial interests, some 135,000 hp in different parts of Norway, all of these to be applied to the manufacture of nitrates from the atmosphere. Most of this power is already developed and will be taken into demand during this year for manufacturing purposes.

Extensive tests have been carried on during the past year with large furnaces taking 4000 hp, and these have been so successful that wherever possible this size will be used.

The Norwegian power conditions are somewhat different from those that most generally exist in this country, since power can be obtained there at a very low figure. This will naturally influence to some extent the financial showing.

Niagara Falls, South Ontario. John Woods Beckman.

The Problems of Tungsten Concentration.—I.

BY H. C. PARMELEE.

The mining and milling of tungsten ores has been a growing industry in Boulder County, Colorado, since 1900. In the early days of gold and silver mining in this district the tungsten minerals were unrecognized, and they received various names from the prospectors and miners. In general they were regarded as forms of iron, and were considered without value. In 1900, however, the first production of tungsten occurred, amounting to 40 tons of ore valued at $3,216. Since then the production of high-grade ore and concentrate has increased annually, except during a depression in 1907-8.

The output for 1909 amounted to 1100 tons of high-grade ore and concentrate, valued by the United States Geological Survey at $396,000. The total output in the United States for the same period was 1607 tons, valued at $559,500. From this it will be seen that the Boulder County production is by far the most important in this country, amounting in 1909 to about 70 per cent of the whole. In 1910 the output from Boulder was 1530 tons of high-grade ore and concentrate, valued by the State Bureau of Mines at $736,700. During the same period the output of all other metals in the county was valued at $161,589.

The tungsten minerals in the Boulder field are the three iron-manganese tungstates, ferberite, hübnerite and wolframite, and the calcium tungstate, scheelite. Ferberite predominates, but scheelite occurs in many mines, as is shown by the occurrence of lime in the analyses of ore and concentrate—there being no other source of lime worthy of consideration. These minerals are of high specific gravity, scheelite being 6 and the other three ranging from 7.1 to 7.5. They are generally soft and friable. The ferberite occurs in three rather well defined forms which, however, frequently grade into each other. These are: (1) Well-crystallized crusts and layers covering the surface of rock fragments and cementing them together; (2) massive granular ore occurring in more dense seams; and (3) the highly silicious ore in which ferberite occurs in fine grains scattered through the gangue.

In breaking ore in the mine the highest grade is sorted and shipped without preliminary milling. At present this grade must average 40 per cent or more tungstic acid. The lower grades, ranging from 3 per cent to 15 per cent tungstic acid, are concentrated by wet methods. Two grades of concentrate are recovered and shipped; the first contains 60 per cent or more tungstic acid, and the second ranges from 35 per cent to 40 per cent. The former is usually recovered as coarse concentrate and the latter as slime.

The stability of the tungsten industry depends, of course, on the uses for the metal; and as these are not extensive and the demand is limited and variable, there is always the possibility of intermittent operation of mines and mills. It is safe to say that were it not for the large companies operating in the Boulder district the industry would have a more variable career than it now enjoys. At the present writing there is a large tonnage of unused concentrates in the Eastern market, as well as at the mills, with no immediate demand in sight. The principal use of the metal is still confined to the manufacture of tungsten steel for high-speed tools. Minor quantities are used in dyeing textiles, glazing pottery, coloring glass and paper, making bronze powder and weighting silks. The manufacture of tungsten electric lamps creates a slight demand, but as 1 ton of high-grade concentrate will suffice for the manufacture of 15,000,000 to 18,000,000 lamps the quantity used is not important.

The reader who may be interested in other details of the Boulder County tungsten field, such as general and economic geology, mining, technology, ores, etc., is referred to the Colorado Geological Survey, which has published a complete report,[1] from which some of the foregoing information has been abstracted.

[1] "The Main Tungsten Area of Boulder County," R. D. George, State Geologist, Boulder, Col.

Location of Mills.

The town of Nederland is centrally located with respect to the tungsten mills of Boulder County. It is reached by stage, 18 miles from Boulder, or by railroad to Cardinal station and thence by stage for 2 miles. At Nederland is the mill of the Wolf Tongue Mining Company. At present this company is running its mill on a custom basis, sampling and paying for the tungstic acid content according to a graduated scale of prices. On ore ranging from 1 per cent to 30 per cent WO_3 the price paid per ton increases with each per cent tungstic acid contained, and above 30 per cent WO_3 the price per pound of tungstic acid also increases with each additional per cent. This system of outright purchase of ore has superseded the former custom of milling the ore at a certain price and returning the concentrate recovered and is generally more satisfactory. It insures better work on the part of the mill and should make a better return to the miner.

At Cardinal, 2 miles west of Nederland, is the Cardinal mill of the Primos Chemical Company. This plant for the past eighteen months has been retreating the tailings made during four years' previous run, when the methods of saving were not as efficient as at present. The Lakewood mill of the Primos Chemical Company is located at Lakewood, about 2½ miles northeast of Nederland. It is the newest and most attractive of the mills in the district. The company's own ore is treated at present, although in the past a great deal of custom ore has been treated.

Between Cardinal and Nederland is the mill of the Alton Mining & Milling Company, a small plant for company or custom work. On Beaver Creek, about 2 miles southeast of Nederland, are two small mills, one belonging to the Tungsten Mining & Milling Company, treating company ore, and the other owned by Smith & Ardourell, at present retreating the old tailings of the Wolf Tongue Mining Company. The Clarasdorf mill on lower Middle Boulder Creek is idle. In addition to the regular milling there is an effort in a small way to roughly concentrate some of the old mine dumps, either by screening and saving the fine mineral or by hand jigging. Any material obtained in this way is sold to the mills.

Difficulties of Concentration.

The concentration of tungsten ore presents difficulties not encountered with other base minerals. This is particularly true with some of the methods now in vogue. It is believed, however, that all of these obstacles can be largely overcome by applying principles and methods that have proved successful in dressing other ores. Excessive sliming is one difficulty, the cause of which is well set forth in the report of the State geologist, previously mentioned.

"Ferberite is a rather soft mineral with one perfect cleavage, and generally one or more prominent partings. As a result the mineral is extremely friable even in the massive and massive-granular forms. Much of the ferberite was deposited as aggregates or loosely arranged crystals and crystal grains, forming crusts over the surfaces of rock fragments. One crust succeeded another until in many places the opening was filled. In other places the cavities remained open, but the walls were lined in the same manner. The crystal grains composing these crusts average not more than ⅛ in. in length and about 1/16 in. in diameter. In much of the ore where the crust is broken the crystal grains are easily separated from one another. When to this ready crumbling of the mass is added the extreme friability of the grains and crystals themselves it is easy to understand the excessive sliming. The finer parts of the slime form an almost impalpable mass which, when stirred in water, gives it an inky appearance, and the water remains turbid for ten days to two weeks. To save these slimes is one of the difficult problems with which the tungsten millman has to contend.

"Another serious problem is the successful treatment of the highly silicious ores. In almost all the tungsten mines there is a certain amount of highly silicious ore, consisting of minute

grains of ferberite in a matrix of chalcedonic quartz or hornstone. The percentage of ferberite varies widely. Outside of certain limited areas this type of ore is fortunately not very abundant and rarely amounts to 20 per cent of the product. Various methods of treatment have been tried, but none has given entirely satisfactory results. Even very fine crushing leaves a large part of the ferberite with particles of quartz attached. In concentrating, these grains consisting of quartz and ferberite will be disposed of according to their specific gravity. Those in which quartz is largely in excess will go with the tailings and ferberite will be lost, while those in which ferberite is abundant will go with the concentrates and help make a low-grade product."

Corroborative testimony on the difficulty of treating some of the ores is given by Mr. Victor G. Hills[2], who was connected with the Colorado Tungsten Corporation during 1905-7. This company treated custom ores, and at that time was doing the best work in the district. Careful records were kept and improvements were made wherever posssible. From a list of twenty-one tests on custom lots the following are given as showing the extremes and average in percentage saving:

Lot.	Mine.	Crude Ore, % WO_3	Concentration, Ratio.	Concentrates, % WO_3	Saving, %
4	D	8.07	8.69	62.17	88.90
3	C	6.07	181.81	40.37	5.03
15	E	9.51	8.93	62.59	75.11

In amount of tungstic acid in the crude ore Lot 3 compares favorably with average lots, but owing to its physical qualities it was a failure from a milling point of view. Mere assays would indicate that it was as good as any other ore, but milling tests proved its worthlessness.

The percentage of saving made in the Boulder County mills is largely a matter of conjecture in many instances. The smaller mills practise no regular control, and the larger concerns prefer to make no statement of their saving. Undoubtedly the efficiency is greater than it was four or five years ago, and as the best mill then operating made a customary saving of 75 per cent to 85 per cent, it is probable that the claims of 90 per cent saving may be realized in present practice. There is little loss in coarse tailing where that portion is allowed to go to waste, and the main loss is in the slime tailing. The statement is made that coarse tailing will assay as low as 0.1 per cent WO_3 and that slime tailings from the best mills contain as little as 0.30 per cent to 0.40 per cent WO_3.

The mills are of small tonnage capacity. Four of them can treat about 5 tons in twenty-four hours, one of them 24 tons and another 40 tons in the same period. With two exceptions they are built especially for tungsten work and are new. The two old mills are remodeled gold mills, and, while not as conveniently arranged as the newer plants, they are doing good work. A notable feature is the large table area required per ton of ore treated, which naturally reduces the tonnage capacity. In no instance is screen sizing being practised with a view to eliminating coarse material which might prove worthless. In view of the friable nature of the tungsten minerals it would seem that sizing should prove worthy of investigation, and certainly classification should be more widely practised.

The stamp is a time-honored crushing device, and finds favor in some of the tungsten mills. Some of the considerations just given, however, indicate that the stamp is out of place in crushing tungsten ore. The minerals are friable and of high specific gravity. When crushed in stamps such material tends to slime excessively, both by reason of natural properties and the fact that it remains too long in the mortar. When this preliminary step has been taken it follows that elaborate provision must be made for saving fine mineral.

Another fact that indicates the wrong use of the stamp is that much of the mineral occurs in a condition which would permit of its liberation by coarse crushing. But, as a matter of fact, a screen analysis of table concentrates made from stamped ore will show about 60 per cent passing a 150-mesh screen and almost 80 per cent finer than 100 mesh. Modern ore-dressing practice would indicate some such method as follows on the average coarse-grained ore: Crushing by jaw crusher and rolls; screen sizing and jigging; regrinding jig tailing, with subsequent classification and table concentration; regrinding table middling, followed by slime concentration on belt machines, and, finally, treatment on canvas if found necessary. Screening might be combined with the classification in such a way as to eliminate a portion of coarse stuff carrying no value.

There is but one mill in the district that is successfully using the jig, and the results are satisfactory to the operators. Other plants are considering the use of coarse concentration, and a second mill will shortly go into commission on this basis.

There are, therefore, two distinct methods of treatment in vogue in the Boulder tungsten field. The first involves comparatively fine crushing with concentration on tables, vanners and canvas plant—the latter being a very important feature and extensive in area. The second includes stage crushing, jigging, concentration on tables and vanners, with a canvas plant as a minor adjunct. The adherents of each method claim an excellent saving, which is not questioned, but the advocates of stage crushing and coarse concentration claim a less expensive plant and cheaper operation. A further claim in favor of coarse crushing is that a greater percentage of high-grade concentrate is recovered, and, since the value of the concentrate is in direct proportion to its grade, it would seem rational to recover as much mineral of high grade as possible and reduce the percentage of fine or low-grade concentrate.

But even with coarse crushing, and every effort made to reduce the quantity of slime, it is noticeable that considerable fine mineral is produced which may float away, even from the boxes receiving concentrates. One of the striking features of all the mills is the provision made for catching this fine floating mineral. Settling boxes of wood or cement are provided, and in one instance the water escaping from the settler passes through a cocoa matting filter. In this connection the idea suggests itself that the use of a coagulant might be beneficial. It is known that fine tungsten concentrate will not settle perfectly clear for several days. The simple expedient of adding a small quantity of lime water will show how quickly the solution will be clarified. On a laboratory scale the mineral thus treated will settle and leave a clear solution in a few minutes. Experiments on a large scale might show that the addition of lime solution at the time of fine grinding would be beneficial in coagulating the mineral and aiding in its recovery. Or, the addition of clear lime water to the concentrate settling boxes might prevent the loss of fine and floating mineral after it had been separated from the gangue. Other salts than lime, which are of proved value in coagulating colloid matter, might be experimented with to advantage.

Magnetic separation has been applied successfully to tungsten ores, but it is not in vogue in the Boulder field. The Boulder ores are remarkably free from sulphides of iron, lead and zinc, cassiterite, arsenopyrite, etc., which usually call for magnetic separation, and this probably accounts for the non-use of expensive magnetic treatment.

The Bingham & Garfield Railway, connecting the mines of the Utah Copper Company at Bingham with its mills at Garfield, Utah, is about 20 miles long. It is laid with 90-lb. rails, has a maximum grade of 2.5 per cent and a maximum curvature on two curves of 10 per cent. The total length of tunneling on the line is 4795 ft. in four tunnels of the following lengths: 682 ft., 754 ft., 1280 ft. and 2079 ft. There are three steel bridges of the following heights and lengths: One across Carr Fork 690 ft. long and maximum height of 190 ft., one across Markham Gulch 640 ft. long and 220 ft. high and the third across Dry Fork 670 ft. long and 185 ft. high. In addition to serving the company's own mines and millls it will haul ores from other mines at Bingham to the smelter at Garfield.

[2] *Proceedings*, Colorado Scientific Society, Vol. IX., pp. 135-158.

Desiccation of Air by Calcium Chloride.*

BY FELIX A. DAUBINE AND EUGEN V. ROY.

The authors have already described elsewhere the principles upon which this process is based, with special reference to the blowing of blast furnaces with dry air and their investigations have been conducted with the object of that application in view. On publicity being given to the process certain manufacturers approached the authors with the desire of putting it into practice. They yielded to these requests and undertook to make investigation of the appliances necessary for drying large volumes of air. An appliance of this nature has been installed at the Differdange Steelworks, Luxemburg, where it is now in normal working.

General Principles of the Process.

It is unnecessary to reproduce here the discussions, theories and calculations which were published in 1909 in the *Bulletin de la Société de l'Industrie Minérale* (Vol. 11, pp. 397-452, 479-541), and of which a summary has appeared in the *Journal of the Iron and Steel Institute* (1910, Vol. 1, p. 613). Nevertheless, it appears to be advisable to recapitulate briefly the general principles upon which the process is based.

It is inherent in the very nature of bodies that every substance the solutions which, when highly concentrated, possess very low vapor tensions, should be, for that very reason, a hygroscopic substance, and capable of serving for the extraction, up to a certain state of equilibrium, of the moisture contained in atmospheres possessing higher vapor tensions than those of the solutions in question. The word "solution" should, in this connection, be understood in its most general sense—a solution of a given strength of concentration may be solid or liquid, this being a matter of temperature—and it will be seen later that one of the characteristics of the process which the authors are about to describe is the maintenance of the solutions obtained in this solid "phase."

The more rapidly equilibrium is attained, the more suitable is the substance for playing its part as a desiccating agent. A highly hygroscopic body would, therefore, be one which, at a given degree of concentration, would possess very low vapor tensions, differing greatly from the vapor tensions of the air to be desiccated. This is precisely the case as regards chloride of calcium and certain of its hydrates, the very low vapor tensions of which permit of obtaining rapid desiccation. In this connection the properties of calcium chloride have been known for a very long period.

It should also be noted that every external influence which is capable of lowering the tensions of the hydrates favors, by this very fact, the desiccation; and as for any given hydrate these tensions vary with the temperature, and in the same proportion, it follows that the lowest temperature possible should always be selected. Further, as the vapor tensions increase in proportion as the molecules of water absorbed render the dilution of the hydrates greater, it becomes equally necessary always to maintain the solution at a sufficiently high degree of concentration.

Consideration of these principles, taken together with numerous experiments, has enabled the authors to determine the economic conditions of the evolution of calcium chloride during the course of its periods of hydration and regeneration. It has, to begin with, been found necessary to avoid making the concentration higher than $CaCl_2 + H_2O$ for the two following reasons: First, that the regeneration of hydrates of greater concentration than $CaCl_2 + 1H_2O$ requires a very large number of calories, so that, despite the very low vapor tensions of these hydrates, they are slow in coming into equilibrium for reasons which are not altogether apparent, but probably depend on a special physical condition. In other words, the $CaCl_2 +$ less than $1H_2O$ hydrates are "tardy" in their operation. Secondly, it has been found that during the absorption of water vapor a dilution of about $CaCl_2 + 8H_2O$ should not be exceeded. This hydrate becomes liquid at a temperature of 15°, whereas all hydrates of higher concentration remain solid at this temperature. Briefly, the details of the process are as follows:

A layer of broken calcium chloride, the smaller pieces at the bottom and the larger pieces at the top, rests on a sieve. Within the mass of the calcium chloride, and in its lower part, is submerged a spiral grating, consisting of pipes for the circulation of water. The volume of air to be dried is made to traverse the mass of calcium chloride from top to bottom, the draught being created by a fan. The absorption of the water by the calcium chloride leads to an evolution of heat, and this heat is carried away contemporaneously with its production by the water circulating within the spiral.

Without this indispensable precaution the temperature of the mass of calcium chloride would rise, the tensions of its hydrates would rise likewise, and the chloride mass would cease to act efficiently. The hydration of the calcium chloride should be arrested when the outside pellicle of the broken pieces commences to liquidate. If it were continued the mass of the calcium chloride would be completely melted, and it would be impossible to regenerate it on the spot.

Further, the fact of its being liquefied indicates that the calcium chloride has formed a hydrate whose dilution is already very considerable (more than eight molecules of water at 15°), and that, as a result, its very high vapor tensions render it unfit for effective desiccation.

At this point it becomes time to regenerate, and to do this it is only necessary to attain the temperatures between which that hydrate best known for this purpose is formed, namely, $CaCl_2 + 1H_2O$, which is produced between 175° and 235°. During the progress of this regeneration it is necessary gradually to raise the temperature in such a manner as constantly to maintain the hydrates in their solid phase.

Suppose, for example, that, after the passage of the air to be dried, the formula of the exterior hydrates is, at a given moment, $CaCl_2 + 4H_2O$, a hydrate which melts at about 40°. Owing to the circulation of the water, which has maintained the temperature of 15°, this hydrate has remained solid; if at the moment of regeneration the temperature be suddenly raised to above 40°, the pellicle corresponding to the above formula melts, breaks down, and is wasted.

It is necessary, therefore, to proceed progressively according to a law which has been recognized in practice and may be deduced in theory from the curve of the liquid and solid phases of the different hydrates of calcium chloride.*

It is necessary also in the course of this regeneration to be careful not to exceed the temperature of 235°, above which the "tardy" hydrate $CaCl_2 + 1H_2O$, of which mention has been made above, is formed.

It may be seen from the foregoing that it is possible to effect the regeneration of the calcium chloride with sources of heat of comparatively low grade, and that it is possible to employ with this object the waste fumes which occur plentifully in all metallurgical works.

The regeneration of the calcium chloride having been effected by this systematic warming, it is necessary in order to render it again fit for the complete absorption of water vapor to cool it completely. This cooling is quickly attained by a rapid circulation of water in the pipe system.

It goes without saying that if the air to be dried were directed on warm calcium chloride no desiccation would take place, because, with the chloride of calcium at such a temperature, the hydrates even if highly concentrated might possess very high vapor tensions. Hence the intermediate period of cooling after regeneration.

When the temperature has returned to that of the average

*A paper read before the Iron and Steel Institute on May 11; slightly abstracted.

Bulletin de la Société de l'Industrie Minérale, December, 1909, p. 497.

environment, the chloride of calcium has regained all its hygroscopic properties, and is capable of desiccating afresh the new volumes of air.

The above constitutes a qualitative summary of the different phenomena which takes place during the evolution of the desiccating substances. It may now be asked, from the commercial point of view, what figures are involved in regard to the relative volumes of air to be dried and of the drying body; to the rate of passage of this air through the body; to the efficiency of the desiccation obtained, and to the amounts of heat to be employed for the regeneration and the time required for the cooling. The replies to these different questions will show at once whether the process is one of commercial application or not. The figures are as follows:

A mass of calcium chloride of 240 kg spread out in a layer of 24 cm in depth, on a square meter of surface, will desiccate 300 cu. m of air, percolating it from above downward per hour for four hours under average conditions of 15 grams of moisture per cubic meter. Without the water circulation the temperature of the chloride of calcium mass would gradually rise to the vicinity of about 40°; the circulation of the water prevents the elevation of the temperature beyond a few degrees, such, for example, as 4 or 5.

In order to regenerate the calcium chloride from its hydrates, calculation shows that it is necessary to expend about 7.500 calories per kilogram of water deposited. In practice it suffices to circulate from below upward—that is, in the opposite direction to the air current—warm air or fumes devoid of dust at gradually increasing temperatures from about 30° to 200° during a period equal to half of the length of time of the passage of the air current—that is, for the example given above for two hours.

Naturally, the circulation of the water is arrested during the circulation of the fumes or of the warm air used for the regeneration. The cooling, owing to the water circulation, does not take any longer time than that of the regeneration. In order to be on the safe side, the authors have always allowed that, given three stages of the evolution, it is necessary to have a triplicate apparatus, so that at any given moment there should always be:

An apparatus at work (desiccation of the air);

An apparatus in regeneration (the circulation of the fumes or of warm air); and

An apparatus in process of cooling.

With regard to the percentages of desiccation obtained they have been deduced already in the calculation. The table below gives the vapor tensions of various compounds of calcium chloride, with water at a temperature of 15°, showing the lower limits which it is possible to attain:

Compounds.	Temperatures.	Pressures in Millimeters of Mercury.	Relative Tensions.
$CaCl_2 + 4$ to $6H_2O$.....	15.5	2.47	0.192
$CaCl_2 + 2$ to $4H_2O$.....	15.0	1.47	0.116
$CaCl_2 + 1$ to $2H_2O$.....	15.0	0.64	0.050

What is called relative tension in the fourth column is nothing less than the state of hygrometric equilibrium which would be attained in a case of repose with indefinite masses of chloride of calcium in presence of finite volumes of air.

Given that the instance under consideration involves a movement of volumes of air and of finite masses of the desiccating substance, these figures themselves can never be reached, although it is always possible to approximate to them. Indeed, experiment has shown that it is possible in a well-watched and carefully conducted operation to attain a degree of desiccation such that the hygrometric condition of the desiccated air will be for the average period of passage but 10 per cent to 15 per cent of its saturation at a temperature of 15°, which corresponds to tensions of water vapor of 1.2 grams to 1.9 grams per cubic meter of air. The foregoing figures have clearly demonstrated in the preliminary experiments that the commercial desiccation of air is possible by such a process. At the present moment it is an accomplished fact.

Earlier Process of the Desiccation of Air by Calcium Chloride.

Fryer Process.—The earliest process that can be mentioned is that of Fryer at Coleford, which made its appearance in 1880. It involved employing the chloride in the most simple form by making the air pass over broken pieces of calcium chloride, collecting the water which settled and regenerating partly in special boilers, and carrying it subsequently to complete igneous fusion.

The inconveniences of this process are at once apparent. If it be sought to liquefy the calcium chloride, there are at once obtained extremely high dilutions, with very high vapor tensions. The tension of the equilibrium itself rises, and finally there is left, in the air to be dried, more moisture than is desired.

This is, however, not the chief inconvenience. The Fryer process allowed for no cooling of the mass during the desiccation, so that the temperature would rise very rapidly, and the calcium chloride became gradually inefficacious.

Finally, the calcium chloride being regenerated in special appliances, a whole series of manipulations had to be provided for—the withdrawal of the liquids, concentration, pouring, solidification, cooling and breaking. The latter operations, taking place on the cold chloride of calcium, are particularly difficult, as it is necessary to take special precautions, and to operate out of contact with air. Indeed, chloride of calcium being a waste product and having in itself no value, it might be affirmed that a chloride thus regenerated would cost practically as much as calcium chloride prepared for the first time. It may be pointed out finally that Fryer, not being aware of the peculiarities of the "tardy" hydrate $CaCl_2 + O$ to $1H_2O$, carried regeneration as far as igneous fusion, which is a disadvantage from the point of view of the activity of the chloride, and requires the expenditure of a considerable amount of heat. The authors are not aware that this process was ever applied commercially.

Cremer Process.—This process dates from 1885, and is illustrated in Fig. 1, in which *A* is a desiccating chamber, in which are arranged the shelves upon which are placed the materials used for the desiccation, sulphuric acid, calcium chloride, or other analagous substances.

B is the air inlet; B^2 is the outlet tube. Inlet *B*, instead of being connected with the open air, may be connected with the discharge vent of a fan, in which case the air may be forced into *B* instead of being aspirated from B^2.

Regeneration can take place on the spot. For this purpose the shelves have a double bottom for the circulation of hot air, or of waste gases to heat the calcium chloride placed upon them.

This process certainly evinces considerable progress, inasmuch as it involves regeneration of the moistened calcium chloride on the spot. It is still, however, exceedingly incomplete. Cremer made the air pass above the layer of calcium chloride instead of making it traverse the layer. The contact is thus less intimate. There is formed in the upper portion an exceedingly moistened layer, the tensions of which gradually increase, as much because of the increasing dilution as because Cremer had not foreseen the necessity of cooling.

Cremer did not indicate either the temperatures of the necessary regeneration, nor the limits within which it was necessary to keep; he provided for the heating of the mass by double-bottomed chambers, which was bad, as the particles in contact absorb all the heat and have a tendency to melt, while the topmost particles are insufficiently heated. Regeneration, in order to be certain, as the authors have shown, requires to be gradual and to be easily regulated. This can be readily attained by the circulation of hot gases through the mass, as the authors have provided for.

Elsner Process.—This process is recent, and dates from 1906. Furnace gases are led on the chloride of calcium, undergoing regeneration in a lower drum. The chloride of calcium, which is assumed to have been regenerated, is raised by means of a mechanical device to an upper drum, in which the air to be dried circulates. Here again no cooling is provided, for although the method of regeneration adopted leads to the supposition that Mr. Elsner wishes to maintain the solid phase

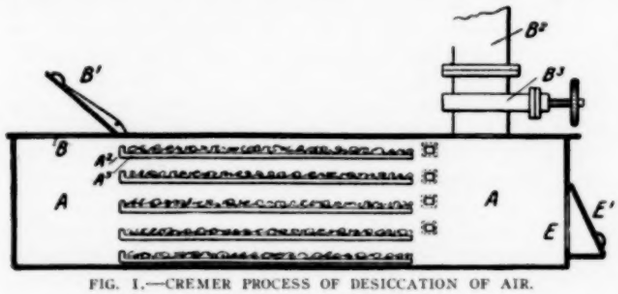


FIG. 1.—CREMER PROCESS OF DESICCATION OF AIR.

for his solutions, he has, nevertheless, neglected the safest method to attain that end—namely, the cooling during the hydration of the chloride.

To any one who has used chloride of calcium, and made it undergo many regenerations, it is a well-known fact that the lumps not only become regular sponges, which are more and more fitted to absorb moisture, but also that their cohesion becomes destroyed little by little, and that these lumps, which very readily resist the passage of air or of gases, pulverize at the slightest shock.

It is very probable that in the Elsner process it would become impossible to maintain the chloride of calcium in lumps, and that the particles would become smaller and smaller and more and more easily liquefiable.

Finally, Mr. Elsner charges his warm regenerated chloride directly into the topmost drum without first cooling it, which is absurd.

Harbord Process.—There are several Harbord processes, dating from 1907, 1908 and 1909. These processes are highly ingenious. Inert substances, such as heat, pumice-stone, all, generally speaking, porous bodies, are impregnated with a concentrated solution of calcium chloride. Regeneration likewise takes place on the spot. The principal object pursued by Mr. Harbord appears to be the retention of such chloride of calcium as might have become liquefied, and the prevention of its flow.

Granting that Mr. Harbord operates with solutions concentrated at high temperatures—above 200°, for instance—the particles retained by impregnation may well be equally active as the peculiar masses which intervene in the process described by the authors. Either one condition or the other must, however, obtain: either the volumes of air to be desiccated are large in relation to the small mass of the chloride of calcium impregnating the inert bodies, in which case there are quickly obtained liquid solutions of high vapor tension, and consequently but little effective, or else the masses of the chloride are, in relation to the volumes of air, in the proportions which the authors have themselves calculated, and the necessity arises for having appliances of enormous dimensions, the quantity of chloride being small in proportion to the quantity of inert matter.

Mr. Harbord has likewise made no provision for cooling, and this cooling would, besides, be difficult to secure, inasmuch as it would have to be carried out simultaneously on the inert bodies which serve to support the active agents. In the last resources, the presence of large masses of foreign matters would be the cause of a heavy expenditure of heat during regeneration, as it would be necessary to raise their temperature as well to the temperature of regeneration.

It would equally be a cause of embarrassment during the intermediary cooling down between regeneration and fresh hydration, as such bodies, which possess, generally speaking, a low calorific capacity, which makes them very slow in heating and causes the absorption of much heat, also require a very lengthy period during which to cool down, which would prevent the calcium chloride being put rapidly into use again.

Mr. Harbord has provided for the use of chloride of iron to prevent the formation of the oxy-chloride. This precaution is superfluous. With chloride of calcium, which had been in daily use during three months, and had undergone over a hundred successive regenerations, the insoluble residue was exactly the same as that of the same calcium chloride on its delivery from the factory.

Other patents are in existence. The bulk of those to which the authors have been able to obtain access resemble one or other of the systems referred to above.

Operation of Process at the Differdange Works.

The problem at the Differdange Works was to desiccate the whole of the air required for the blowing of the blast-furnace of 150 tons per twenty-four hours, in order to realize for this blast-furnace the economies described by Mr. Gayley in his experiments of 1904 and in those he had undertaken since.

The appliances were designed by the firm of Paul Wurth & Co., Luxemburg. Drawings showing two forms of apparatus are given in Figs. 2 to 8.

In the first design the blast is introduced by a central well and distributed in layers by means of the openings leading to different superposed reservoirs. On emerging from the latter it is collected in an annular chamber, where it is led to the place where it is to be utilized.

In a second form of appliance, on the contrary, the blast arrives by way of the annular chamber and is collected in the

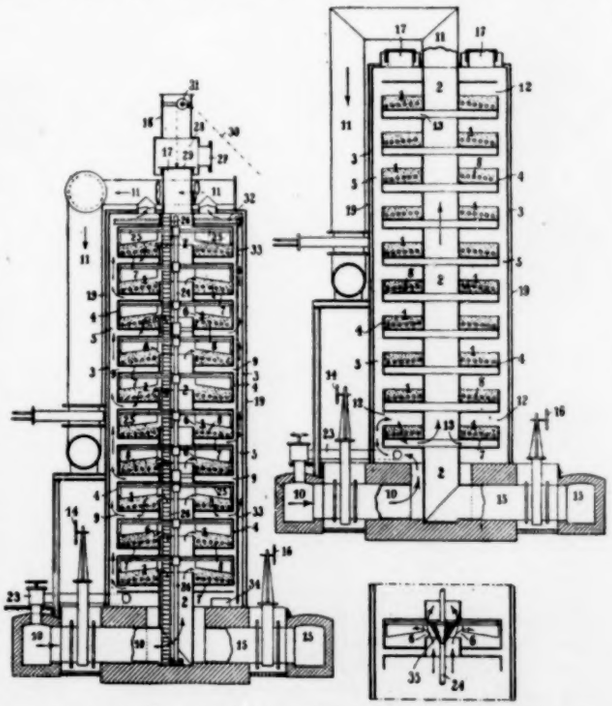


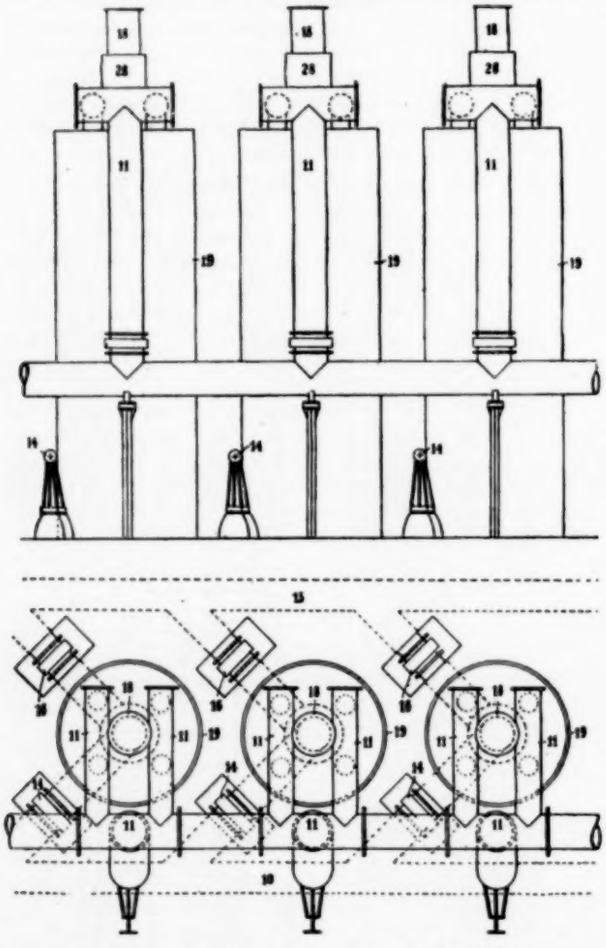
FIG. 2, 3, 8.—DETAILS OF PROCESS OF AIR DESICCATION.

central well after traversing the layers of chloride of calcium and is led thence to the place of utilization.

In order to be able to regenerate the chloride of calcium on the spot, the central well (or the annular chamber) is connected with a second pipe, admitting air or hot gases. The arrangement of the fans is such that it is possible, at will, either to pass the air to be dried, or the gases which are to serve for the regeneration, through the apparatus, or else to isolate it completely during the cooling down.

Figs. 2 to 8 show the two forms of design. Fig. 2 shows, diagrammatically and in longitudinal section, the first type of design, and Fig. 3 the second type. Figs. 4 and 5 show in elevation and in plan an installation composed of several drying apparatus; while Figs 6 and 7 show, in longitudinal section and in plan, the details of the spiral cooling arrangement. Fig. 8 shows details of the means for distributing the air current.

According to Figs. 2 and 3, the apparatus, which has a cylin-



FIGS. 4 AND 5.—DETAILS OF PROCESS OF AIR DESICCATION.

drical shape, is composed of a certain number of compartments identical with each other (1), a central shaft (2), a cover (3), which forms, with the external shell (4), a number of compartments, and the annular chamber (5).

Each compartment (1) has a circular inlet orifice (6) and a grid (7), on which the chloride of calcium rests. An outlet (9) is provided below the grid.

Towards the end of the desiccation the air is drawn by means of the fan across the main lead (10) into the central shaft (2), and distributed to all the compartments. It traverses the chloride of calcium from above downwards, depositing its moisture; it then escapes into the annular chamber and reaches the conduit pipe (11), which takes it to the place of utilization.

As is well known, the chloride of calcium warms during the desiccation, and this must be rectified by the cooling. It is with this object that water is made to circulate in the spirals provided in each compartment. These spirals are arranged to have a certain inclination to provide for their being rapidly emptied. The joints are effected in a special box securing a water-tight separation for the chloride of calcium, with a discharge vent in case the joint becomes defective. The chloride is thus completely protected against such an event occurring.

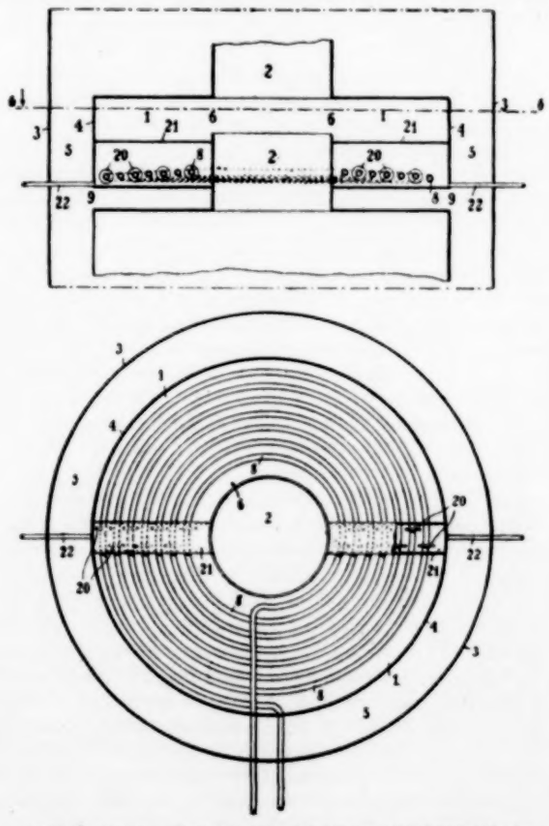
When the chloride of calcium has absorbed the requisite amount of moisture, the apparatus is cut off from the inlet and outlet of air by the slide valve (14), and communication is made by means of the slide valve (16) with the main (15) which conducts air or warm gases.

As the temperature should only be increased gradually, the fan aspirates cold air at the same time as it takes in the gases, to allow the temperature being regulated. By diminishing the amount of air admitted and increasing that of gas the temperature is gradually raised to 235°.

When the calcium chloride has been regenerated the apparatus is first cooled by means of a cold current of air entering the chamber (5) by the conduit pipe (23) and escaping by the outlet orifice (17) and then water is introduced in the spiral cooling coils until the temperature has been sufficiently lowered. The apparatus is then ready to desiccate the air-blast anew.

In the installation shown in Figs. 4 and 5 three apparatus have been assumed to be employed, of which one dries the blast, while in the second the chloride of calcium is being regenerated and the third is undergoing cooling. During the period of the passage of the blast, or at the end of the period of cooling, it is possible to obtain access to the appliance, either by the central shaft by means of the ladder shown in Fig. 2 or to the annular chamber by the ladder (33), running on rails at the top (32).

In the application of the process at Differdange, which is shown on Fig. 9, the conditions require the drying of 30,000



FIGS. 6 AND 7.—DETAILS OF PROCESS OF AIR DESICCATION.

cubic meters of air hourly. The apparatus, constructed in accordance with Figs. 2 to 8, possess the following features:

Total area presented to the passage of the blast, 100 square meters per apparatus.

Number of compartments, 10.

Depth of the layer of calcium chloride in each compartment, 24 centimeters.

Apparent density of the chloride of calcium, 1.0.

Weight of chloride of calcium contained in each apparatus, 24,000 kilograms.

Weight of chloride of calcium contained in all three appliances, 72,000 kilograms.

Cooling surface of the spirals in each apparatus, 170 square meters.

These appliances have been designed to work in the most unfavorable conditions—that is to say, to remove, during the summer months, 15 grams of moisture per cubic meter of air during a period of four hours.

At the time of writing this paper the appliances have been working normally for six weeks, but as the season is the end of winter and as the moisture in the air is not very large, it has been found unnecessary to make as many reversals as were contemplated.

Each apparatus receives the blast for 6 to 8 hours. The air which contains 6 to 8 grams of moisture before its passage only contains 1 to 1.5 grams per cubic meter on emerging from the apparatus, and this figure remains practically constant from the commencement to the conclusion of the period.

Regeneration requires four hours for its completion and is carried out by means of the waste smoke gases from boilers and from Cowper stoves. These gases, cleaned to the extent of 0.4 gram per cubic meter, pass directly through the mass of chloride. The temperature is regulated at 30° to commence with, and thereafter gradually raised in conformity with a certain ascertained law up to about 200°. In the summer the temperature will be carried to 275°.

Cooling takes three hours.

It is interesting to compare these results of drying the blast with those which are capable of being given by the freezing method used by Mr. Gayley.

The authors reproduce below a table of the vapor tensions of ice and the amounts of moisture which saturate a kilogram of dry air at temperatures below 0°:

Temperatures. Degrees.	Vapor Tensions in Millimeters of Mercury. Per Cent.	Weight of Water Vapor in Grams in Saturated Air per Kilogram of Dry Air. Per Cent.
0	4.60	3.80
—1	4.27	3.50
—2	3.95	3.20
—3	3.66	3.00
—4	3.39	2.80
—5	3.13	2.60
—6	2.89	2.40
—7	2.67	2.20
—8	2.46	2.00
—9	2.27	1.85
—10	2.09	1.70

From the above table may be seen to what temperature the air must be cooled to obtain results identical with those obtained at the Differdange Steel Works.

At 15° an amount of humidity of 1.5 gram per cubic meter corresponds with the content of 1.22 gram per kilogram of dry air. This figure, as may be seen, is well within the limits of the foregoing table. To obtain this result, calculation shows that it is necessary to cool the air effectively to —15°, which requires even considerably lower temperatures in the refrigerating tubes.

This shows the advantage which the employment of calcium chloride has over that of refrigerating machines. While with the latter system cooling to —5° leaves a minimum of 2.60 grams per kilogram of dry air, the use of the calcium chloride regenerated under the form $CaCl_2 + 1$ to $2H_2O$ and maintained in sufficient excess at 15° allows of a degree of moisture as low as 0.5 gram per kilogram of dry air being attained.

During the winter months, during which the refrigerating system is practically inoperative, particularly in Continental European localities, chloride of calcium allows of exceedingly low figures being attained—that is, distinctly less than 1 gram per cubic meter of air.

With regard to the blowing tests on dry air now being undertaken at the Differdange blast furnace it is impossible for the authors to give any details at present. The installation is too recent for the true value of the economies realized to be stated, but it is now quite established that the manufacture of dry air need present no difficulty whatever.

The installation has cost a little less than one-quarter of what would have been the cost of an installation for desiccation by means of refrigerating machines. One man for the day shift and one for the night shift are sufficient to handle the apparatus, which is of the most simple description. The expenses of working are thus greatly reduced.

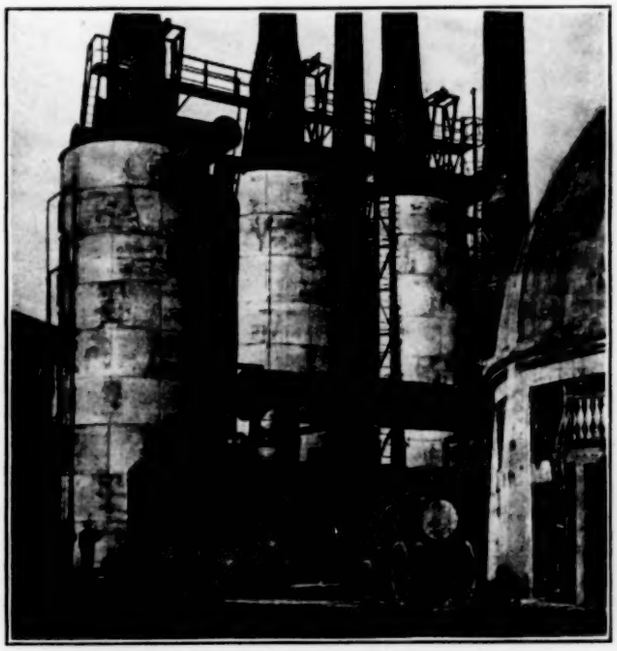


FIG. 9.—AIR DRYING BY CALCIUM CARBIDE AT THE BLAST FURNACE PLANT OF DIFFERDANGE STEEL WORKS IN LUXEMBURG.

The authors state, in conclusion, that similar apparatus are under investigation, not only for other metallurgical works, but also with a view to their application in other industries that require drying operations carried out at low temperatures, as, for example, for the desiccation of chemical or pharmaceutical products, india rubber, resins, gums, gelatin, albumen, glue, aniline colors and various organic bodies. It may thus be seen that there is a vast field open for the commercial production of dry air.

Aubone, France.

The Detection of Willemite by Ultra-Violet Light is used at Franklin Furnace by the New Jersey Zinc Company for the examination of tailings. It has been found that ultra-violet rays cause willemite to show a peculiar luminous glow, which enables one to examine in broad daylight tailings containing this mineral. The method used was described by Mr. George C. Stone, chairman of the New York Section of the Mining and Metallurgical Society of America, as follows: The apparatus consists of an arrangement to produce an arc between iron electrodes, which is rich in ultra-violet rays. Every barrowful of tailings is examined by the foreman, and if he believes that the material contains more than 2 per cent zinc it is returned for retreatment. The mill foreman becomes quite expert in examining material in this manner, but he is checked by assays. The discovery that willemite could be detected in this manner was made in 1896 and is due to Dr. Baskerville and Dr. Kunz.

The Melting Point in Its Relation to Alloying Capacity.

By Dr. Hans Goldschmidt.

The opinion is often expressed that if a metal has a high melting point it is difficult to produce with it an alloy of uniform composition. When scrutinized, however, this opinion is found to be unwarranted. When I first introduced the

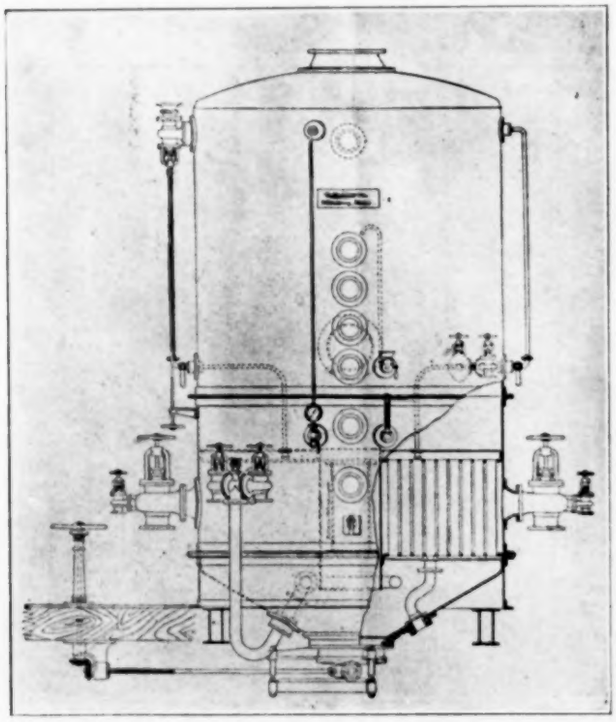


FIG. 36.—GERMAN VACUUM PAN WITH SHORT VERTICAL HEATING TUBES.

aluminothermic metals, such as chromium, manganese, ferrotitanium, etc., into commercial practice, I had to overcome considerable trouble, since metallurgists would tell me that the melting point of these carbon-free metals or alloys was higher than that of the usual brands then on the market which contained carbon, and that the higher melting point of my metals would interfere with their use.

Meanwhile, practice has long shown that carbon-free chromium, in spite of its melting point of 1550° C. (2822° Fahr.) alloys easily with molten steel. The same is true for carbon-free manganese with a melting point of 1245° C. (2273° Fahr.) and this manganese alloys especially easily with copper; in fact, more easily than manganese containing carbon.

In recent years titanium is often added to steel. Ferrotitanium made for this purpose by the aluminothermic process has a melting point of 1330° C. (2426° Fahr.) and alloys easily with liquid steel.

In the preceding paragraphs I have stated the melting points although they are not of practical but only of scientific interest. The following notes will make this clear:

An alloy is nothing but a solution. For instance, an alloy of zinc and copper is called a "solid solution." Zinc has a melting point of 419° C. (786° Fahr.), copper a melting point of 1098° C. (2008° Fahr.). Now let us try to alloy lead instead of zinc with copper; lead has a melting point of 330° C. (626° Fahr.), hence much lower than zinc, yet our efforts of producing an alloy of copper and lead will not be successful. It is simply a fact that lead does not dissolve in copper.

Among the most refractory metals known we have platinum with a melting point of 1775° C. (3227° Fahr.). Yet platinum alloys with tin and lead at a temperature of the melting points of these two metals. Many a young chemist has been sorry to make the experience that when by accident a small amount of these metals got into his beautiful new platinum crucible a hole was formed when it was heated. The fact is simply that the high melting point of platinum does not protect the crucible from the dissolving action of the metals mentioned.

Since, as stated before, an alloy of two metals is nothing else but a solution of one in the other, we are justified from a scientific standpoint to consider it as analogous to the solution of common salt in water. We know that common salt dissolves easily in water; the melting point of common salt, which is at 820° C. (1508° Fahr.) is without any significance for the process of solution.

From what has been said it is clear that for their practical purposes metallurgists have simply to take care that they employ metals or alloys which will dissolve. And it may become necessary to use intermediary alloys which dissolve more easily in the metallic bath. For instance, a high-percentage manganese-copper alloy is made from pure manganese, while instead of pure titanium ferrotitanium is used.

This carbon-free ferrotitanium, containing 25 per cent of titanium, is used now quite extensively as an addition to steel for making rails and rolls. This ferrotitanium made by the aluminothermic process dissolves very easily and uniformly in the molten steel. It has a small intentional content of aluminium, as this increases its solubility.

The experience of metallurgists has shown that as a rule carbon-free metals dissolve more easily than metals containing

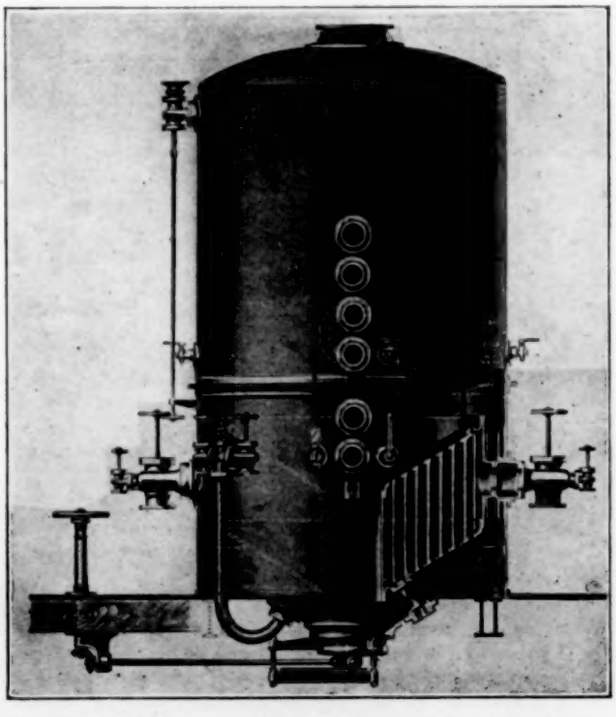


FIG. 37.—GERMAN VACUUM PAN, WITH INCLINED REMOVABLE HEATING SYSTEM AND VERTICAL TUBES.

carbon, since the latter contain carbides—that is, chemical compounds with carbon. It is true that the melting point of the carbides is lower, but as was shown before, this has nothing to do with the solubility, or what is the same, the alloying capacity. The situation is practically the same with ferrovanadium as it is with titanium. For titanium and vanadium it is advantageous to use an intermediary alloy, because only very small quantities of these metals are used.

When a metallurgist has the choice between metals made by different processes the main point of importance is to use starting materials of absolutely uniform composition. Only if he is sure that they will not vary in composition he can get good reliable results. This requirement is fulfilled for metals made by the aluminothermic process on account of their method of production, since they are not made in small quantities, drop by drop, as is the case in the arc or melting zone of electric furnaces, but are produced from a uniformly fused charge. Uniform products are obtained as a result of the uniform composition of the thermit mixture. On the other hand it is very difficult to maintain uniformly the melting zone in an electric furnace, and for this reason it is practically impossible to obtain an absolutely uniform composition in electric furnace alloys, such as ferrochromium, ferrotitanium and ferrotungsten, as especially the carbon content varies considerably.

Essen-Ruhr, Germany.

Evaporators and Vacuum Pans.

BY B. VIOLA, M.E.,
Member American Society of Mechanical Engineers.

(*Continued from page* 311.)

Already in many German vacuum pans short heating pipes have been adopted, but if so arranged the belt or frame of the pipes are generally inclined and the heating tubes are in a

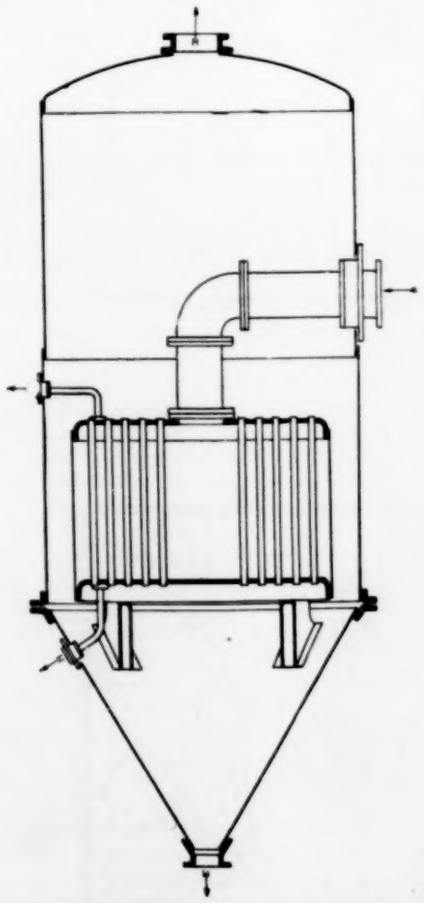


FIG. 37a.—PASSBURG VACUUM PAN, WITH REMOVABLE HEATING CHEST.

vertical position. These heating systems can be made steady or removable; the last ones being only hooked in and can be exchanged any time (Figs. 36 and 37). An American-made vacuum pan with removable heating chest can be seen in Fig. 37a of the Passburg system. This is another specialty, in so far as the heating steam does not enter on the side of the heating tubes, but on the top of the heating chest, which gives the steam a better opportunity to heat the pipe surfaces more equally and by that a better evaporation can be effected. The main point is, however, that the steam pipe carrying the steam into the steam chest should always be immersed in the liquor.

The most highly recommended shape for the inclined heat-

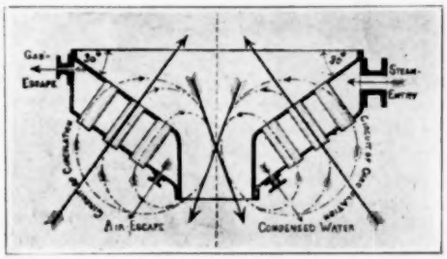


FIG. 38.—PRINCIPAL OF THE INCLINED HEATING TUBE SYSTEM.

ing tube system is shown in Figs. 38 and 39, and is the foundation of Stade's express vacuum pan. This most modern pan consists of one or more heating systems which are arranged in the following manner. The calandria consists of conical parallel plates, which are fixed like a double funnel and contain a central circulation pipe inside and a cylindrical ring outside. The heating tubes are fixed at such an angle that their central lines form a kind of parallel or central radiatiform (Fig. 38).

The steam enters at the upper part of the calandria, the condensed water escapes at the lowest point, while for air and gas separate pipes are arranged to insure highest efficiency of the system. The tubes are short and wide. They deliver even the most concentrated masses up to 5 per cent water to the top. The tubes being set inclined, the masses have to get out sideways and are forced to circulate sideways in or outside of the system.

There are only about three or four supports required for one system, consequently there are no complicated stays to hinder

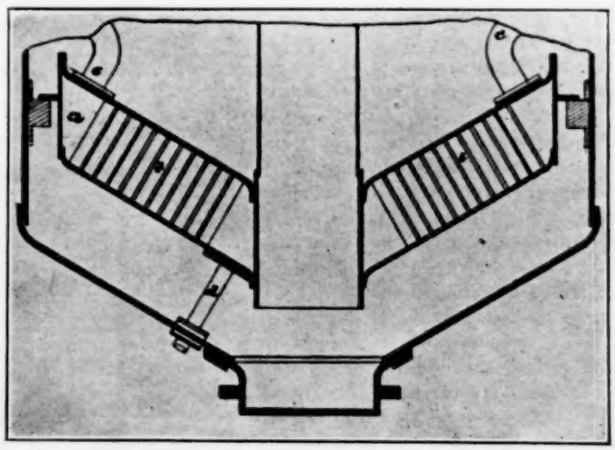


FIG. 39.—REMOVABLE INCLINED HEATING-TUBE SYSTEM.

the full circulation which is bound to take place in a methodical forced system. (Fig. 39.)

One of the chief points of the express vacuum for practical engineers is the easy access to all parts of the system. The tubes can be reached from outside and inside, so that in case of any leakage repairs can be effected in the most simple manner.

The fundamental principle on which the circulation system in general use now (in combination with short inclined express tubes) is based upon may be explained as follows:

The construction consists of a double or false shell fixed at certain intervals over the calandria (Fig. 40). The shells separate the large interior space of the pans in such a manner that a forced but natural circulation takes place from inside up (through the inclined express tube system) to outside down (the space between false shell and pan shell). All the superheat taken up through the tubes goes in form of vapor straight to the condenser and the so cooled masses possess again the necessary low temperature to get the highest transmission of heat when coming again into contact with the calandria tubes.

The natural law states that in two communicating tubes filled with liquids of different specific gravities the levels must be of different height. Now the express circulating system contains inside above the express heating system a mass-cuite mixed with the vapor generated. Outside, however, between the inner and the outer shells the mass-cuite remains free of any vapor as by action of the vacuum pump the vapor generating from the masses escapes at once to the surface and goes to the condenser. Consequently the specific gravity of the mass-cuite outside is higher than the specific gravity of the masses above the calandria. This causes the inside masses to overflow over the inner shells.

In addition to this circulation, however, the action of the heating tubes of the calandria has to be taken into account. These tubes serve as a powerful suction pump, drawing the mass-cuite from the bottom and delivering it through the express tubes to the inside shells system, helping in this way to form a strong circulation in the most simple way, doing away with all stirring and all movable parts inside the pan.

All kinds of steam can be used with the new express circulation pan, and a pressure of 7½ lbs. is sufficient even for large pans with high heads of mass-cuite. With steam of 10 lbs. pressure careful experiments with a large pan gave an evaporation of 15 lbs. water per square foot heating surface.

The materials used for the construction of vacuum pans are copper, iron, lead or lead-lined sheet iron. The heating coils or tubes are copper in most cases on account of its better heat-conducting power, or lead and lead-lined copper, according to the materials which has to be concentrated.

Pans made of copper are usually round, while those of iron are cylindrical or half cylindrical with flat bottoms.

Each vacuum pan, similar to the last effect of the evaporators, is provided with a foam catcher, juice catcher, and has an amply large diameter connection pipe with the condenser.

For the evaporation of certain acid liquors, lead or lead-lined sheet iron is

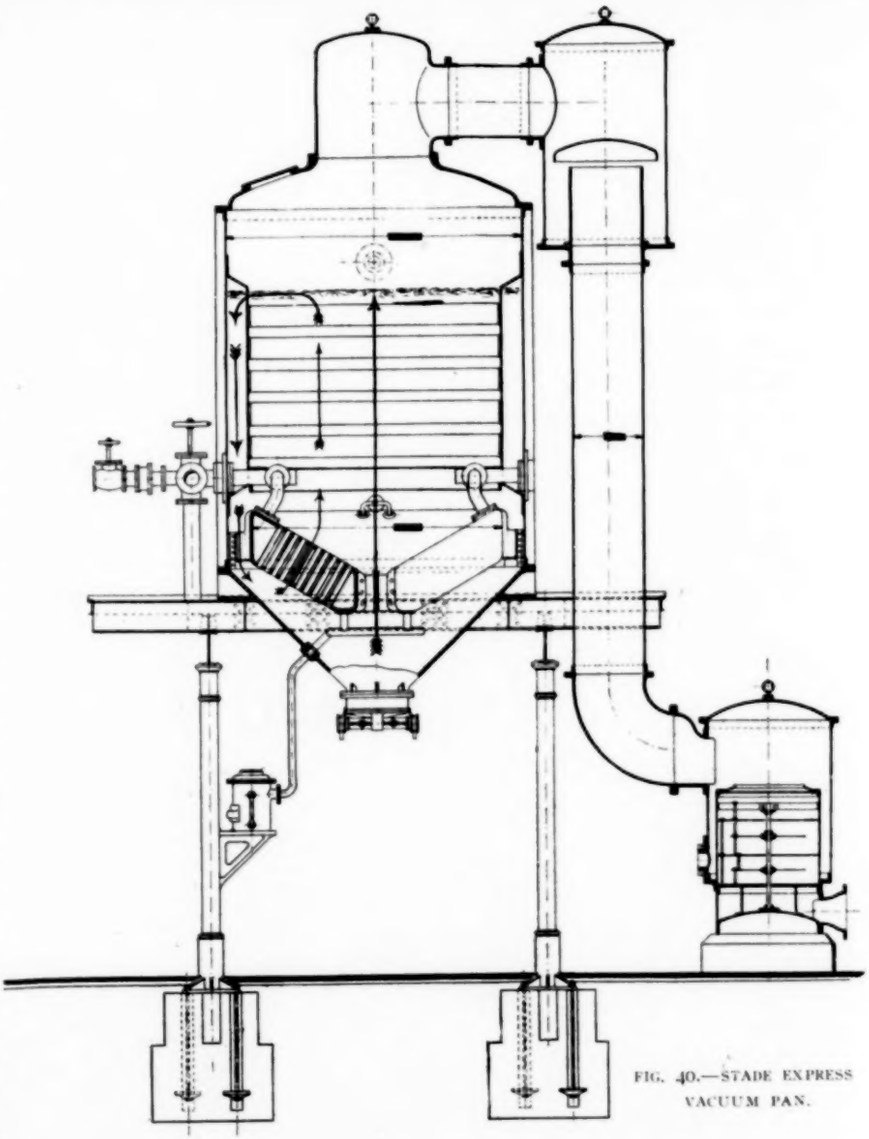
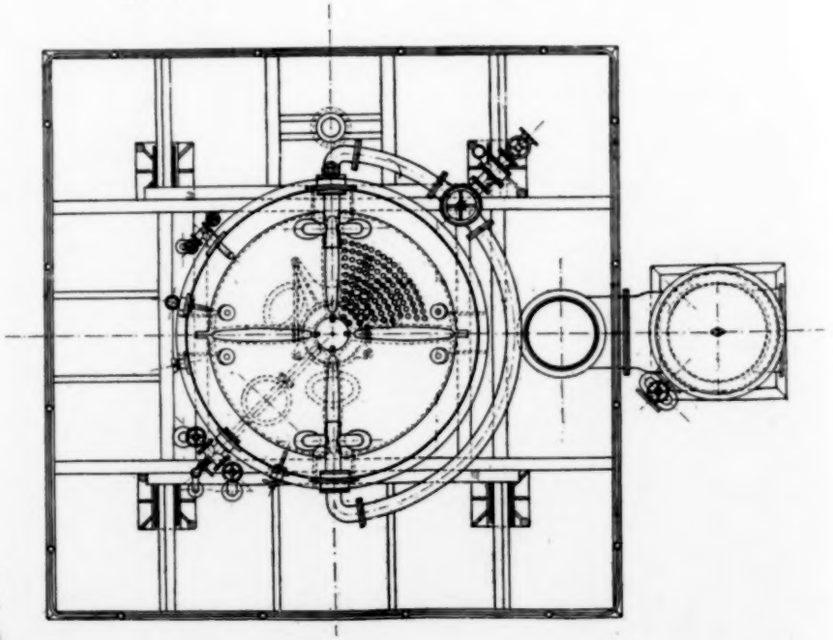


FIG. 40.—STADE EXPRESS VACUUM PAN.



the material mostly used for vacuum pans. If lead is used, then on account of the thickness, the whole pan has to be made of lead casting. Naturally the heating coils were made of lead also, in spite of their poor heating capacity, because no other material can stand certain acids. An entire lead vacuum pan with lead heating coils can be seen in Fig. 41.

Lead vacuum pans had neither foam nor juice catchers, and so it proved necessary to catch the acid particles, which were carried along with the vapors. For this purpose cylindrical cast-iron vessels or so-called overflows were connected with the vapor discharge of the vacuum pans, so that the acid particles could settle there and only the vapors passed on to the condenser. In order that the iron should not be destroyed by the acid particles some neutralizing material had to be used. A complete equipment of such a plant is shown in Fig. 42 vacuum pan, overflow and condenser. However, these overflows were destroyed very quickly and after a while were given up entirely and foam and juice catchers were used.

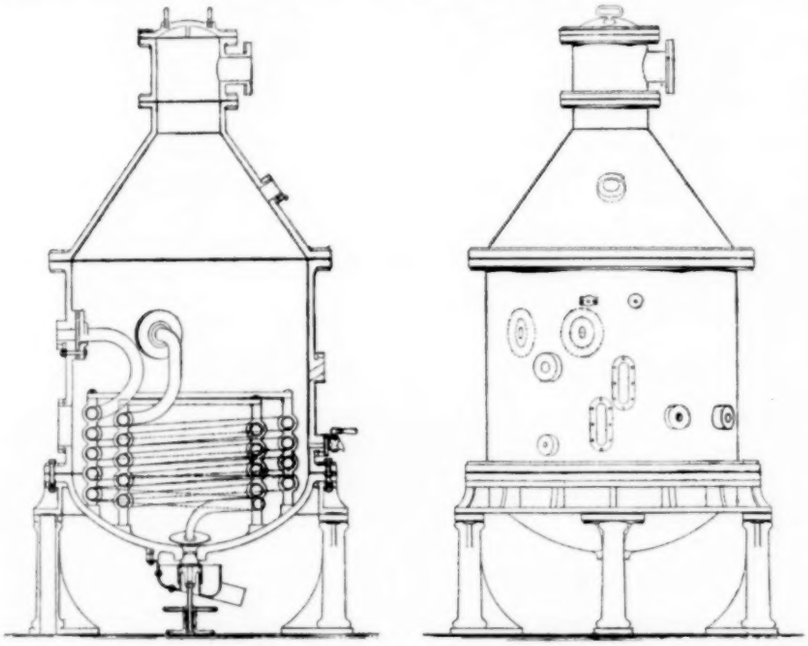


FIG. 41.—CAST-LEAD VACUUM PAN, WITH HARD-LEAD HEATING COILS.

Lead-lined sheet iron for vacuum pans and lead-lined copper coils for heating the acid liquids can be used with great success. Lead-lined sheet iron can be handled more easily than lead casting and be made much lighter in weight, which makes it preferable for the construction of vacuum pans. Lead-lined copper coils have a larger heating efficiency than lead; higher steam pressure can be used and little repair is required.

Fig. 43 shows the top of a lead-lined vacuum pan with foam catcher in the dome and a juice catcher in the vapor line, connected to the condenser.

Fig. 44 shows a complete disposition plant of a homogeneous lead-lined double vacuum pan, provided with single lead-lined copper heating coils and catch-all in the dome, through which the vapors pass to the condenser.

For the evaporation of acids the following constructions have to be mentioned also, which are built by the "Kuehnle-Kopp-Kausch" or homogeneous lead-lined system. Very often two separate evaporators are connected with one juice catcher and one condenser. Such an installation is represented in Fig. 45. These two evaporators are equipped with two homogeneous lead-lined copper coils, about 100 ft. long each and 2¾-3 in. diameter, having an evaporating capacity of about 16 to 20 lbs. water per minute. Both evaporators are working together, but as shown any of them can be cut out occasionally.

A very interesting homogeneous lead-lined evaporator is Fig. 46, which in fact is a double effect. Between the first and second effect we see a catch-all, where all the drops carried over from the first pan by the vapors, are caught and the vapor passes to the second effect, where a higher vacuum is obtained. This second effect has a catch-all in the dome, so that as few drops as possible should be carried over into the overflow, which is inserted between the pan and the plainly seen counter-current condenser.

These heating coils have in the first effect 3⅜-in. diameter by 300 ft. length and a heating surface of about 323 sq. ft. A homogeneous lead-lined copper coil for about 85 lbs. working pressure is followed by a lead-coil in the second effect with 3⅝-in. diameter and 323 ft. length. About 17 gal. of sulphate-alum solution are evaporated per minute. The liquor is in the first effect boiled and the vapors from there are used for the second effect, where the cooking process is finished. In this case both effects have to work together.

Another arrangement for an acid vacuum-evaporating plant, something similar to Fig. 45 is shown in the two homogeneous lead-line vacuum pans of Fig. 47. However, each pan has its own overflow and condenser, but both condensers are connected with one water separator and vacuum pump. This arrangement makes it possible that each vacuum pan can work independently from the other. The necessary valves are inserted between the condensers and tail pipe, cooling-water inlet and in the connections of the condensers and water separator.

Each apparatus is provided with a homogeneous lead-lined copper coil and each has a heating capacity of about 162 sq. ft. and concentrates per minute 10 gal. liquor to 2¾ gal. of acid.

It is always preferred to use above-mentioned equipments, because if two vacuum pans are connected to one condenser bronze or lead valves have to be used to cut out one pan. However, bronze is destroyed too quickly by the influence of acids and lead valves also do not last long enough; they get untight in a very short time.

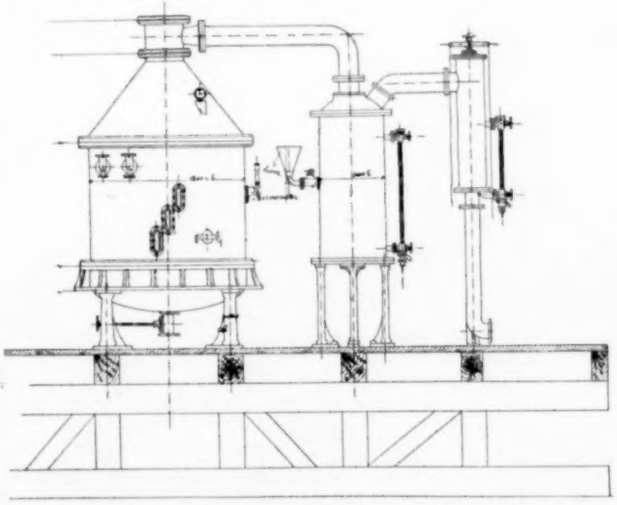


FIG. 42.—VACUUM PAN WITH OVERFLOW AND CONDENSER.

Fig. 48 is a homogeneous lead-lined vacuum evaporator with vertical heating pipes. The steam chamber encloses the outer surfaces of the pipes and the liquor circulates through them

exactly the same way as already mentioned in another part of this paper. To give the liquor a more lively movement and thereby enforce the evaporation, a propeller mixer is equipped in the center of the evaporator, which drives the cooler layers of the bottom always upward, making 150 rev. per minute. The setting of these mixers caused the change of the situation of the vapor outlet. This evaporator has about 280 sq. ft. heating surface and evaporates about 13 gal. of acid liquor per minute. It can be heated with steam from about 65 lbs. pressure.

A very simple construction of a homogeneous lead-lined evaporator with jacket is shown in Fig. 49, having 4 ft. 3 in. diameter, about 52 sq. ft. heating surface and when heated by live steam of about 80 lbs. pressure it evaporates around 4 gal. of liquor per minute. On the top in the center, the vapors are escaping to a condenser.

[This serial is to be concluded in our next issue when vacuum dryers will be dealt with.]

The Control of Thermal Operations and the Bureau of Standards. *

By George K. Burgess, D. Sc.

The name that is generally recognized as standing foremost in the determination of the thermal properties of materials is that of Regnault, but it is perhaps not as generally known that the experimental investigations in heat carried out by him over an interval of some thirty years, beginning in 1840, were initiated and supported by the French Government. The problem that Regnault first attacked was that of the properties of steam, a particularly opportune subject at that epoch, and the title page of his first memoir on the subject reads as follows:

"Account of the Experiments undertaken by order of the Minister of Public Works, and on the proposition of the Central Committee on Steam Engines to Determine the Principal Laws and the Numerical Data which enter into the Calculation of Steam Engines."

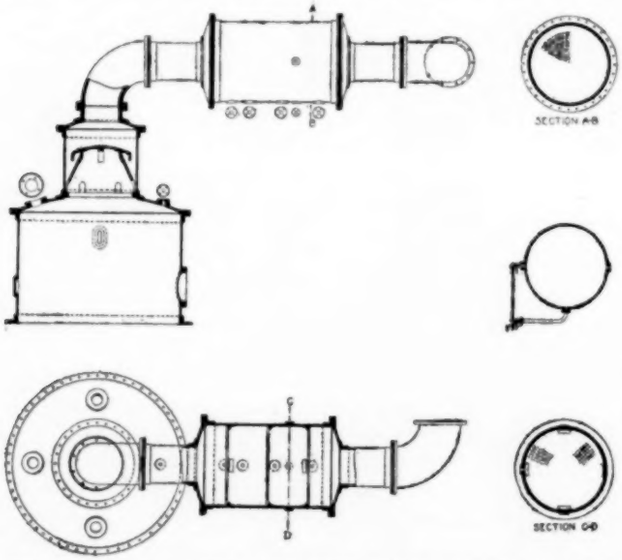


FIG. 43.—TOP OF VACUUM PAN WITH FOAM CATCHER AND JUICE CATCHER.

This was published in 1847, or seven years after the granting of the government subsidy. This delay was in part due to the necessity Regnault labored under of carrying on many researches, fundamental and subsidiary to his main problem, including, for example, the construction of both a temperature and pressure scale, as well as determinations of absolute densities and of specific and latent heats.

The example thus set by France in giving sustained governmental aid for the determination of thermal constants of fundamental importance has been appreciated and followed after many years, it is true, by other countries, several of which have established government laboratories for the carrying out of investigations, not only on the fundamental thermal properties of materials but also, in some instances, with activities covering

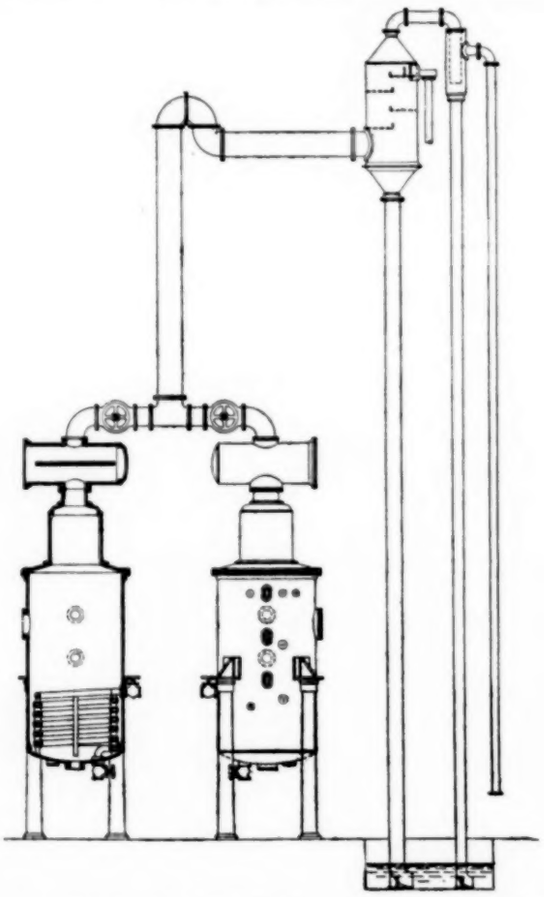


FIG. 44.—HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PAN WITH SINGLE LEAD-LINED COPPER COIL.

the whole range of the physical sciences, including the testing of instruments and materials submitted by the public. Of this type are the German Imperial Physico-Technical Institution (P. T. Reichsanstalt), established in 1887, the National Physical Laboratory of Great Britain, organized in 1899, and the Bureau of Standards founded only ten years ago.

It would take us too far afield to endeavor to cover the whole subject of standardization and its relation to industry, and it will serve our purpose to limit ourselves to the consideration of a domain, that of heat, in which the development of standards has been less rapid and less exact than in many other lines. We shall sketch in some detail the part taken by the Bureau of Standards in the control of thermal operations, with the object of calling attention to an aspect of the question of economic efficiency that is of fundamental importance, namely, the rôle of common units and standards, their establishment, maintenance and availability.

It is a most painstaking matter to set up and maintain a temperature scale, in fact the textbooks state that temperatures cannot be measured and that we can only recognize or locate a given temperature in terms of some other phenomenon that we can recognize or measure. For such reasons, in great contrast to the certainty and permanence of our standards of

*A paper read at the Congress of Technology, April 11, 1911, in celebration of the fiftieth anniversary of the granting of the charter to the Massachusetts Institute of Technology.

length, mass, time, and more recently the electrical quantities, we have had and still have great discrepancies in our temperature scales and all the varied quantitative relations and thermal constants depending on them.

It is eminently fitting, therefore, and particularly necessary that there be a central authority for the establishment, maintenance and distribution of a single temperature scale and the

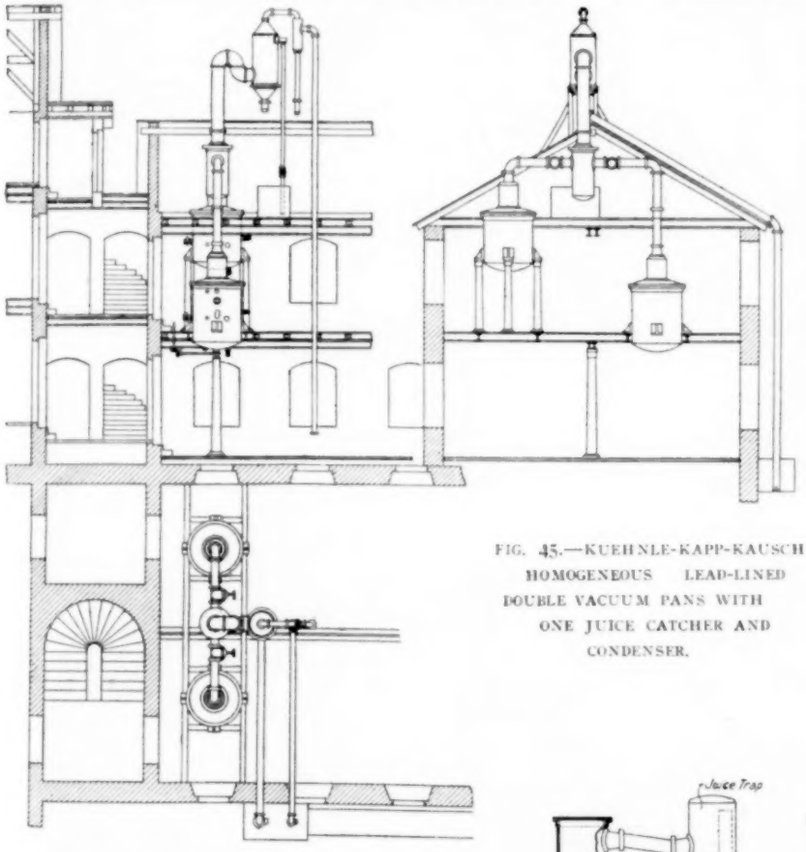


FIG. 45.—KUEHNLE-KAPP-KAUSCH HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PANS WITH ONE JUICE CATCHER AND CONDENSER.

derived heat units. The International Bureau of Weights and Measures, established in 1871 at Sèvres near Paris revising the work of Regnault and others, performs this function in the first instance for the fundamental thermometric interval 0-100° C, while the higher and lower temperature ranges and derived quantities have been left mainly to the various national laboratories since their establishment, as they were previously left entirely to individual initiative. Among the most brilliant examples of the latter are those of Barus in the early eighties, when he was physicist of the Geological Survey, Rowland at the Johns Hopkins, and our own Holman.

Experience has shown, however, that it is quite impracticable for any individual working alone, and engaged in other pursuits as he must be, to maintain and disseminate a temperature scale. Indeed, progress in exact measurements in thermal constants and in all physical constants depending on temperature has often been greatly hampered in the past by the unavailability of any common temperature scale in terms of which results might be expressed. Each investigator was speaking a different temperature language which he could translate into no other. It required, for example, several elaborate, subsequent researches to express the results of Rowland's magnificent work on the mechanical equivalent of heat in terms of the hydrogen scale of the International Bureau. To-day it is neither necessary nor desirable for an independent investigator to try and set up his own temperature scale. Continuity and permanence of thermal standards can only be maintained to advantage by laboratories especially equipped for the purpose, the organization and working of which are not dependent on any individual and the equipment and staff of which are adequate to meet the demands of the public.

Since its creation in 1901, the Bureau of Standards has been actively engaged, in so far as its resources would permit while developing many other lines of work, in serving the United States in the capacity of an advisor on best values and available methods in the many lines of thermal measurements. This activity has necessitated the undertaking of a considerable number of experimental investigations, the carrying on of a very extended correspondence and the execution of many tests of instruments and materials. The bureau does not impose its authority in thermal standards on anyone, and any prestige in this field it may possess must rest on the reasonableness of its suggestions.

The temperature scale of the International Bureau has been reproduced here

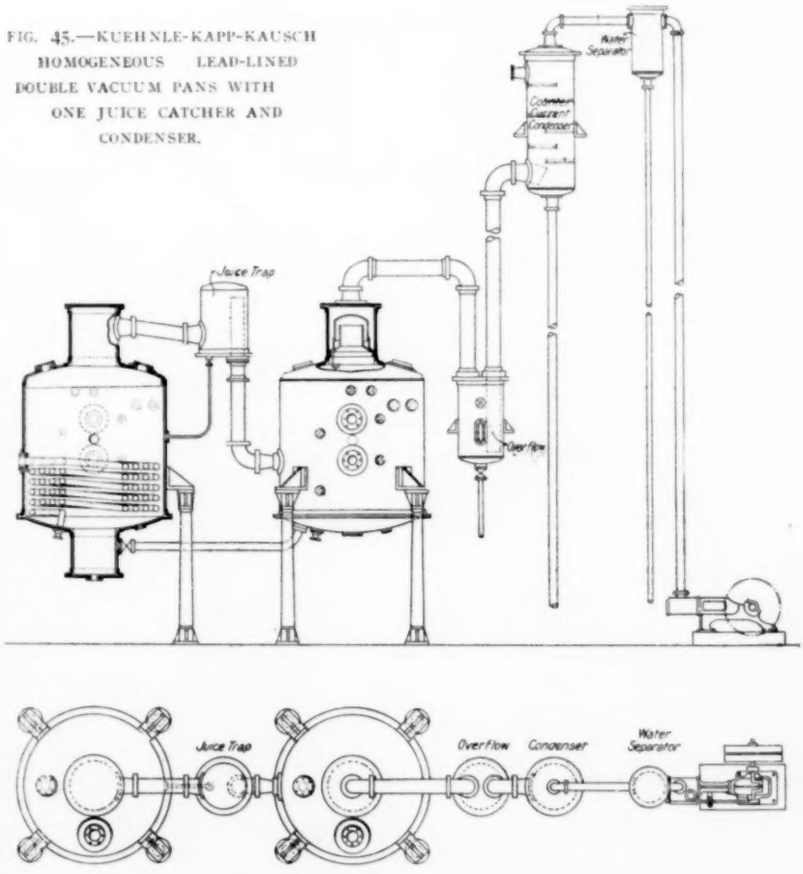


FIG. 46.—HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PANS FOR SULPHATE ALUM SOLUTION.

to 0.°002 C in the interval 0 to 100° C, or to as close as temperatures are known within this interval; and for the remainder of the temperature range, for which there is no international authority, a temperature scale has been constructed which, except perhaps for the very highest temperatures, is in substantial agreement with those maintained

by the German and British laboratories. This temperature scale of the Bureau of Standards is reproducible to about 0.°03 at 500°, 1° at 1000° and 10° at 2000° C and has required for its establishment a great deal of experimental work, and advantage has of course been taken also of similar work elsewhere.

It may be of some interest to consider the relation of this bureau to some of the various bodies with which it comes in contact and mention in particular some of the thermal problems to the solution of which it has contributed.

The bureau is by law constituted the custodian of the standards and is authorized by the organic act creating it to exercise such functions as "the comparison of the standards used in scientific investigations, engineering, manufacturing, commerce and educational institutions with the standards adopted or recognized by the government; the construction, when necessary, of standards, their multiples and subdivisions; the testing and calibration of standard measuring apparatus; the solution of problems which arise in connection with standards; the determination of physical constants and the properties of materials when such data are of great importance to scientific or manufacturing interests and are not to be obtained with sufficient accuracy elsewhere."

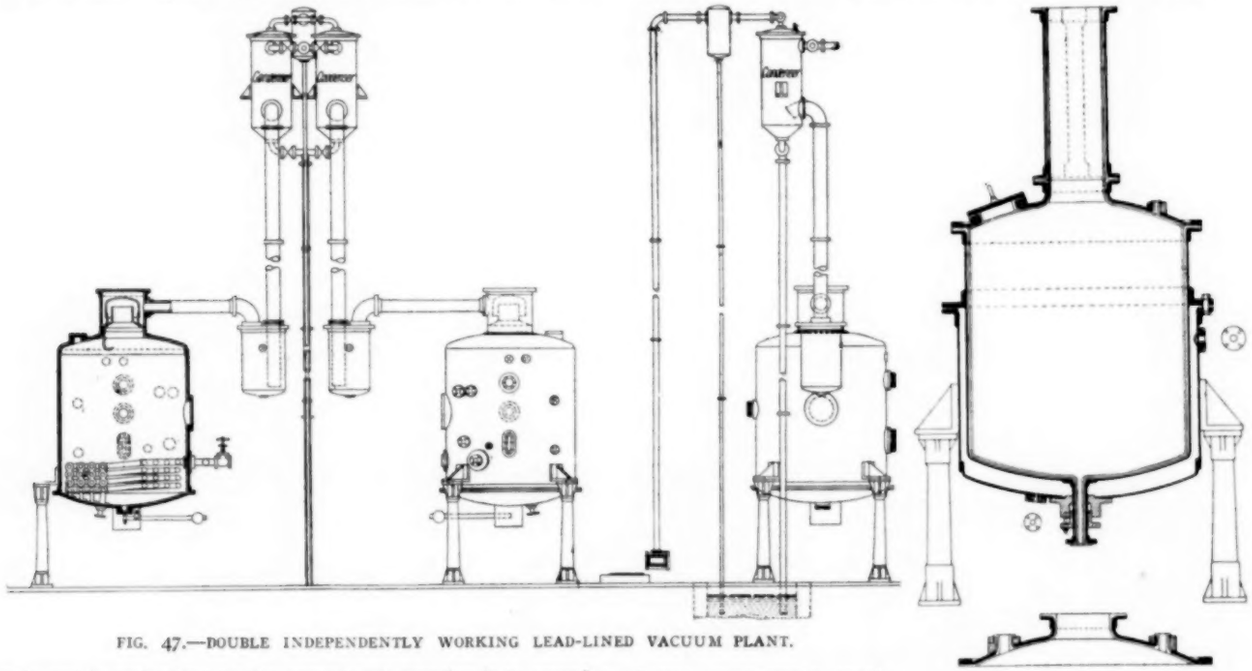


FIG. 47.—DOUBLE INDEPENDENTLY WORKING LEAD-LINED VACUUM PLANT.

Questions arising in the departments of the government requiring advice or decisions on the physical properties of materials are usually referred for solution to the bureau, and this demand for its services and advice is growing at a greatly accelerated rate for tests of various kinds of temperature measuring instruments, for the formulation of specifications, for the purchase of thermal apparatus and of materials on which thermal tests may be made to control their quality, and for the carrying out of specific experimental investigations on materials, processes or methods involved in the solution of some problem in which one or another department of the government is interested.

To mention a few examples among many of the control thus exercised, the purchase of clinical thermometers for two of the departments as well as of the various kinds of thermometers of several of the scientific bureaus is based on specifications drawn up by the bureau and its testing of the instruments before their acceptance, and the knowledge on the part of manufacturers that a bid may be rejected due to the findings of a disinterested, competent authority is not detrimental to the class of instrument submitted in competition for purchase in this way. Experience has proven this practice effective.

The purchase of refractory brick by the Panama Canal Commission is also based on such tests, as well as that of many other materials.

The decision as to whether contested materials are inflammable and, therefore, to be barred from carriage on passenger steamers, is left to the bureau, and such decision sometimes entails an unexpected amount of experimentation.

Investigations of lubricating and illuminating oils and of the types of apparatus used in the testing of them are being carried out with several objects in view, such as the drawing of better government specifications, the furnishing of data for better and more uniform testing methods both in this country and by international agreement. The tests of viscosity, for example, are now on a purely empirical basis, different countries use different instruments and in the United States there are several incommensurable instruments in common use. It is hoped to reduce all such measurements to a common basis.

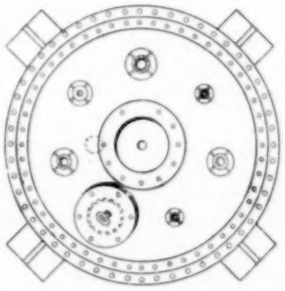


FIG. 49.—HOMOGENEOUS LEAD-LINED EVAPORATOR WITH JACKET.

The bureau has been asked to take part in the work of several international and national societies or committees, usually with the object of establishing by experimentation the necessary conditions or specifications for the carrying out of some method of testing or standardizing materials, methods or instruments. We have not been able to meet all the demands of this kind in the field of thermal operations, but have limited ourselves to some of the problems that are in the most unsatisfactory condition. Besides the oil question, which is very troublesome, the subject of combustion and gas calorimetry has been attacked from the foundation, and although less than two years have been spent on this, we not infrequently receive praiseworthy letters of a slightly ironic cast asking for our final results on this or that phase of the problem. Accepting Regnault's standard, we still have five years to make good on this problem,

however. An outsider does not always realize that no one is more impatient to finish his problem than the worker himself, but where fundamental standards are concerned the worker is bound to be more conservative than his severest critics.

In this calorimetric work it has been necessary, for example, to design and install a complete plant for the production of pure oxygen, new calorimeters, resistance thermometers and accessory apparatus capable of giving results of the highest accuracy attainable at the present time, and the thermal constants must be determined with different sets of apparatus and checked by several methods.

This calorimetric investigation is very far-reaching and involves, for example, the heat values to be accepted as standard in this country for the constituent components of gas burned in every city and town and, therefore, the ultimate basis on which the price of gas as a fuel should be fixed. The heats of combustion which are being determined of those substances suitable for calibrating bomb calorimeters, in a similar way, will define the ultimate units on the basis of which coal and oil fuels should be bought and sold. Concrete calorimetric standards are realized in the form of certified samples of pure materials of known heat value, which are distributed to interested parties. The various types of calorimeter are also being compared, and incidentally to the whole investigation, improved methods of measurement and new instruments have been developed.

To the refrigerating industries we have furnished after elaborate experimentation the correct values of the specific heat of brine at low temperatures, a constant of very great importance to them. There is great need for further systemization of units in this field as well as the better determination of other constants fundamental to refrigeration. The question of these units will be gone over very carefully at the next International Congress of Refrigeration, which meets in this country next autumn.

The relation of the bureau to the manufacturers of precision instruments is very close, manifold, and in general most cordial. When it is remembered that previous to 1901 there was no generally recognized standardizing authority in this country for thermometers and that each manufacturer had his own scale tied up in one or more long cherished instruments, the breakage of which was a real calamity for him and his clients, the necessity for the establishment of a standards bureau is self-evident.

Besides the testing of thermometric instruments, and thereby furnishing all manufacturers with a single scale, uniform everywhere, the bureau, in certain cases, loans its own standards. It has also taken a very active part in the improvement of specifications and the methods and materials of manufacture as well as in testing methods. The resulting improvements in American thermometer manufacture have been such that in 1906 Gehr. Wiebe, the chief of the thermometer department of the Reichsanstalt, published the statement that the German thermometer manufacturers were complaining more and more of the loss of the American market.

The manufacturers of instruments for the measurement of high temperatures have also been aided greatly, and the condition in this field may be described as having been chaotic before the benefits of uniformity of calibration could be realized. Opportunity is also given for trying out and developing new types of instruments, and a great many suggestions have been given manufacturers which have been incorporated in their instruments. The standards of practically every pyrometer and thermometer manufacturer in the country have been tested at the bureau.

The past ten years has witnessed great strides in the development of the manufacture and use of pyrometers. Formerly the question usually asked by an engineer responsible for the control of some process involving change of product or output with varying temperature was "Do I need a pyrometer and is there a reliable one?" To-day he knows he needs one and asks which is the best for his purpose. The bureau carries on a very heavy correspondence with the users of such instruments, endeavoring to furnish them with the information best adapted to each case, although no pretense is made of replying to the often asked question, which is the best pyrometer?

In a similar way, advice has been furnished to universities

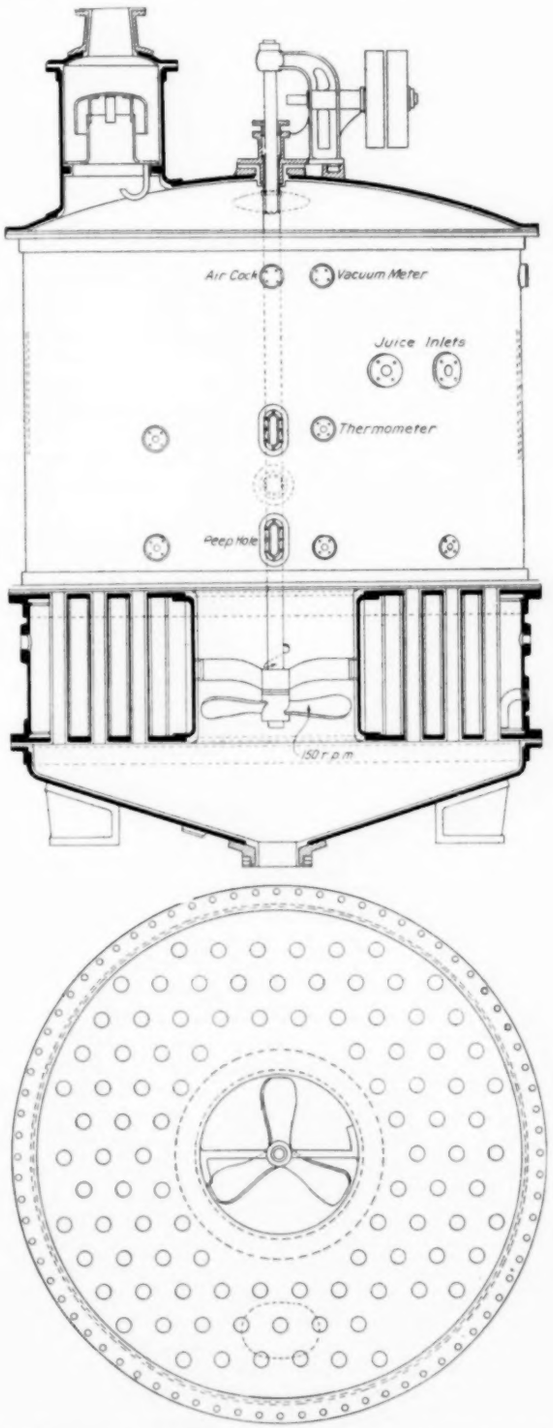


FIG. 48.—HOMOGENEOUS LEAD-LINED VACUUM EVAPORATOR WITH VERTICAL HEATING PIPES.

and technical schools regarding the formation of courses of instruction in heat measurements, the installation and design of new apparatus, and the testing with the highest possible accuracy of instruments to be used by them in numerous experimental investigations.

To still further aid in the dissemination of a uniform tem-

perature scale, preparations are being made for the distribution of samples of pure metals and salts of certified melting points, thus permitting any one to check his own thermometric or pyrometric apparatus in place. As a preliminary to this a careful survey of the reproducibility of the thermal behavior of such materials from several sources of supply has been made.

Among the other investigations under way or in immediate contemplation that have a bearing on the control of thermal operations may be mentioned studies of thermal properties of steels and refractories, including in both instances the purest obtainable materials and the commercial products; the properties of steam with respect to turbine design; the melting points of the elements and of some pure salts and alloys; gas thermometry; the temperatures of incandescent lamps; the corrections to be applied to optical and radiation pyrometers when sighting on steels, bronzes, clays, and so forth; and the comparison of the laws of radiation.

Some of these experimental problems have been undertaken at the suggestion of one or more interested individuals or corporations who often are in a position to furnish materials that may sometimes be difficult to obtain otherwise. Indeed, we are at times in the position of being offered a great many more opportunities of this kind than we can embrace.

Still another function the bureau has occasionally exercised, and one that will probably develop, is to act as referee or court of appeals in case of disputes involving standards, physical properties of materials and the like, and for the interpretation of contracts involving testing methods.

The testing activity of the heat division of the bureau for the past four years ending June 30, 1910, was as follows:

	1906-7	1907-8	1908-9	1909-10
Clinical thermometers.....	8,444	8,395	10,955	13,082
Various thermometers.....	577	798	1,791	1,508
Pyrometers	46	60	31	39
Calorimetric tests.....	....	....	41	58
Oils: Flash and viscosity.....	42	17	159	17
Miscellaneous	9	99	6	45

Several of these miscellaneous tests were of the nature of quite elaborate investigations and most of the pyrometers were manufacturers' standards. Nominal fees are charged for all tests except for the United States or State governments. About one-third of this testing is for the United States government. The bureau makes known the results of its investigations by means of its Bulletin, and its testing methods and specifications are described in Circulars of Information.

With the ever-increasing economic competition and the improvement of scientific methods and instruments as applied to the arts and sciences, the demand for higher accuracy of calibration is also constantly growing. Where, for instance, 0.°01 C was considered an ample precision a short while ago in combustion calorimetry, and 0.°001 C sufficed yesterday, the demand is now for a certainty of 0.°0001 C in temperature differences at 25° C. The meeting of such rigid requirements necessitates oftentimes the application of entirely new methods and instruments, and we are witnessing in consequence the passing of some familiar types, such as the mercury-in-glass thermometer, from the list of instruments of precision.

A most important step is yet to be taken in the unifying of the temperature scales and the values assigned to the various thermal constants in use in the several countries. To-day a pyrometer manufactured in France, calibrated in England and used in the United States comes in competition with a German-made and certified instrument and with one of American make and certification. They will each give a different temperature for a furnace of say 1500° C. We have the example of the unification of the standards of weights and measures as well as those of electricity and photometry, and it is to be hoped that ere long the standards of heat may also be rendered universal.

Washington, March 21, 1911.

The Methods of the United States Steel Corporation for the Technical Sampling and Analysis of Gases.

Copyright, 1911, by J. M. Camp, Chairman Chemists' Committee.

(*Concluded from page 306.*)

2. Blast Furnace Gas.—Before the analysis is commenced, the solutions in the pipettes are brought flush with the capillary tube, while the latter and burette are filled with water. The petcock on either end of the sample tube containing the gas sample (No. 2, analytical apparatus), is filled with water and attached to the rubber tubing, also full of water, leading from the shelf bottle (No. 1), while the other end of the sample tube is attached to the rubber tubing at the end of the main capillary and the sample tube is placed in its rack on the side of the apparatus. The leveling bottle is then placed on top of the apparatus, the pinchcock on the water supply, the lower, then the upper cock on the sample tube, and the three way cock on the capillary tube are opened: the water from the shelf bottle forces the gas into the burette against the pressure from the leveling bottle, which is lowered if necessary, the gas being always maintained under pressure.

About 25 cc of gas are thus forced into the burette, and then expelled through the stem of the three-way cock, all connections remaining as they are. This sample is rejected because it contains the air that was in the rubber connection between the capillary and sample tube. A second sample is forced in until it reaches the base of the burette and about one minute is allowed for the burette to drain and the temperatures to equalize; during this period the sample tube may be disconnected from the apparatus. With the leveling bottle on top of the apparatus, the tube leading from it is pinched between the thumb and finger and the three-way cock opened, the pressure of the gas being thus released. The remaining portion is slowly forced out by releasing the pressure until the meniscus just reaches the zero mark in the burette, then the three-way cock is closed. If the water in the leveling bottle is brought to the same level as that in the burette, the meniscus of the latter should be exactly at zero.

Determination of Carbon Dioxide. The leveling bottle is slowly raised after the cock in the first reagent pipette containing the solution of potassium hydrate has been turned so that the gas will bubble through the solution. If the leveling bottle should not be raised slowly the bubbles which first appear may be lost by being carried into the other compartment of the pipette. When all but about 5 cc of the gas has passed into the pipette the cock is turned to a position for drawing the gas back into the burette, keeping the leveling bottle raised, and the instant the reagent has been expelled from the side capillary of the pipette by the remaining gas the leveling bottle is lowered and the gas drawn back. Thus the gas is forced in and drawn out of the pipette at least three times, the reagent finally being drawn to its initial point in the stem of the capillary. The burette is allowed to drain for about one minute and the reading taken. The difference between this and the initial reading represents the percentage of carbon dioxide in the gas. If the percentage of carbon dioxide is high, to make certain that it has all been removed, the gas should again be passed into the solution of potassium hydrate and a reading taken as before.

Determination of Oxygen. The passing of the residue from the absorption of carbon dioxide into the fuming sulphuric acid for the absorption of the illuminants or olefiant gases is usually omitted with blast furnace gas, as these are either entirely absent or present only in such small quantities that they can scarcely be detected in the usual way. Therefore the residue is at once passed into the second pipette containing stick phosphorus and left there for about one minute to remove the small amount, if any, of oxygen found in blast furnace gas. The difference between the burette reading obtained and that after the absorption of carbon dioxide represent the percentage of oxygen present.

Determination of Carbon Monoxide, Hydrogen and Methane. One-half the above residue is used for the determination of these gases. The last reading when substracted from 100 shows the volume of the residual gases remaining in the burette. One-half this residue substracted from 100 shows the point on the burette the meniscus must reach to reserve one-half the residue. If it is desired to duplicate the explosion, one-half of the residue is reserved in the phosphorus pipette. If not, the leveling bottle is placed on top of the apparatus, the connecting rubber tubing pinched, and the residue to be rejected is slowly forced out through the stem of the three-way cock until the meniscus just reaches the calculated point. The three-way cock is closed and by the application of the leveling bottle the calculated reading should be obtained.

Approximately 45 cc of the oxygen and nitrogen mixture are now passed into the burette, through the permanent rubber tubing connection, and an exact reading is made. If the carbonic acid in the absence of oxygen is low in the sample it indicates a gas rich in combustibles, in such case a proportional increase is made in the amount added of the oxygen and nitrogen mixture. The leveling bottle is raised, the cock leading to the explosion pipette opened, and the gases are passed over and back two or more times to insure a thorough mixture; the gases are exploded, drawn back into the burette and after the lapse of one minute Contraction I is noted.

The residual gas is passed through the potassium hydrate pipette at least three times to remove the carbon dioxide. After the burette has drained a reading is made, which, less the preceding reading, gives Contraction II. The residual gas consists of nitrogen and oxygen, and for the purpose of calculation, the oxygen must be known. About 35 cc of hydrogen are added, an exact reading is taken, and the gases are passed back and forth into the explosion pipette until a thorough mixture is effected. The gases are exploded, drawn back into the burette, allowed to drain and cool and a reading again taken. This reading, less the former, represents the contraction due to the combustion of hydrogen and oxygen. This is known as Contraction III, and one-third of it represents the oxygen present prior to the last explosion.

In the analysis of all gases there should be an excess of hydrogen after Contraction III, hence two-thirds of Contraction III should be less than the hydrogen added; if it is not less oxygen still remains, this is removed by passing the residue into the phosphorus pipette until the oxygen is removed. The contraction is noted, multiplied by three and added to Contraction III; one-third of this total is the oxygen present prior to the last explosion.

The remaining gas consists of nitrogen and hydrogen, which do not enter into the subsequent calculations, as all the necessary data has been obtained for calculating the amounts of carbon monoxide, hydrogen, methane and nitrogen that existed in the original mixture. The reactions which take place in the combustions are as follows: $2CO+O_2=2CO_2$, or two volumes of carbon monoxide unite with one volume of oxygen, forming two volumes of carbon dioxide, there being a contraction of one volume. Therefore, the contraction and the oxygen used are each one-half the carbon monoxide, and the carbon dioxide formed has the same volume as the monoxide. $2H_2+O_2=2H_2O$, or two volumes of hydrogen unite with one volume of oxygen forming water vapor which condenses, the contraction being three volumes. The contraction is then three-halves the hydrogen, and the oxygen consumed is one-half the hydrogen. $CH_4+2O_2=CO_2+2H_2O$, or one volume of methane unites with two volumes of oxygen, forming one volume of carbon dioxide, the water condensing and the contraction being two volumes. The contraction and the oxygen used are each twice the methane and the carbon dioxide has the same volume as the methane.

From these facts the following equations are formed:

Contraction I=½ carbon monoxide+3/2 hydrogen+2 methane.

Carbon dioxide formed=carbon monoxide+methane.

Oxygen consumed=½ carbon monoxide+½ hydrogen+2 methane.

From these three equations a variety of formulæ may be derived for calculating the components of the original mixture. The simplest are:

Oxygen consumed = oxygen added — ⅓ Contraction III.

Hydrogen=Contraction I—oxygen consumed.

$$\text{Carbon monoxide}=\frac{\text{Con. I}+(4\times\text{Con. II})+\text{Con. III}}{3}-\text{oxygen added.}$$

Methane = Contraction II — carbon monoxide present.

As the analysis by explosion was performed on half portion the results must be doubled for expression of percentage. Nitrogen in all cases is obtained by difference, i. e., it is equal to 100 minus the sum of the percentage of the other constituents of the gas.

Example—Data.

	Burette Readings.	
	0.0	
Potassium hydrate.....		11.3% carbon dioxide.
	11.3	
Stick Phosphorus.....		0.0 Oxygen.
	11.3	
	88.7	Residue.
	44.3	½ Residue.
	55.7	Water level.
Add oxygen and nitrogen mixture		46.6 mixture added, 44.4% oxygen.
		20.7 oxygen added.
	9.1	
Explosion		9.9 Contraction I.
	19.0	
Potassium hydrate.....		12.9 Contraction II.
	31.9	
Add hydrogen.....		31.4 hydrogen added.
	0.5	
Explosion		39.0 Contraction III.
	39.5	13.0⅓ Contraction III, or oxygen in excess.

Calculations.

20.7 oxygen added.
13.0 oxygen unconsumed, or ⅓ Contraction III.

7.7 oxygen consumed in explosion I.
9.9 Contraction I.

2.2 hydrogen present.

9.9 Contraction I.
51.6 4 × Contraction II.
39.0 Contraction III.

100.5 total.

33.5 ⅓ total.
20.7 oxygen added.

12.8 carbon monoxide present.
12.9 Contraction II.

0.1 methane present.

Explosion ratio: I, 2.99; II, 1.55

Analysis.	Per Cent.
Carbon dioxide.....	11.3
Oxygen	0.0
Carbon monoxide	25.6
Methane	0.2
Hydrogen	4.4
Nitrogen	58.5
	100.0

Occasionally it is found that the gas when mixed with hydrogen for the second explosion will not explode, due to the fact that, the gas sample being abnormally high in combustibles,

Table of Factors for Correcting the Volumes of Gas to Standard Conditions of 62° F. and 30" Mercury.

Temperature in Degrees Fahrenheit	Barometer in Inches of Mercury.																				
	28.5	28.6	28.7	28.8	28.9	29.0	29.1	29.2	29.3	29.4	29.5	29.6	29.7	29.8	29.9	30.0	30.1	30.2	30.3	30.4	30.5
40	992	995	999	1002	1006	1009	1013	1016	1020	1023	1027	1030	1034	1037	1041	1044	1047	1051	1054	1058	1061
41	990	993	997	1000	1004	1007	1011	1014	1018	1021	1025	1028	1032	1035	1039	1042	1045	1049	1052	1056	1060
42	988	991	995	998	1002	1005	1009	1012	1016	1019	1022	1026	1030	1033	1036	1040	1043	1047	1050	1054	1057
43	986	989	993	996	1000	1003	1007	1010	1013	1017	1020	1024	1027	1031	1034	1038	1041	1045	1048	1052	1055
44	984	988	991	994	998	1001	1005	1008	1012	1015	1019	1022	1026	1029	1032	1036	1039	1043	1046	1050	1053
45	982	986	989	992	996	999	1003	1006	1010	1013	1017	1020	1023	1027	1030	1034	1037	1041	1044	1048	1051
46	980	984	987	990	994	997	1001	1004	1008	1011	1015	1018	1021	1025	1028	1032	1035	1039	1042	1045	1049
47	978	982	985	989	992	996	999	1002	1006	1009	1013	1016	1020	1023	1026	1030	1033	1037	1040	1044	1047
48	976	980	983	986	990	993	997	1000	1004	1007	1010	1014	1017	1021	1024	1028	1031	1034	1028	1041	1045
49	974	978	981	985	988	992	995	998	1002	1005	1009	1012	1015	1019	1022	1026	1029	1033	1036	1039	1043
50	972	976	979	983	986	989	993	996	1000	1003	1007	1010	1013	1017	1020	1024	1027	1030	1034	1037	1041
51	970	974	977	981	984	987	991	994	998	1001	1004	1008	1011	1015	1018	1022	1025	1028	1032	1035	1039
52	969	972	976	979	982	986	989	992	996	999	1003	1006	1010	1013	1016	1020	1023	1026	1030	1033	1037
53	967	970	973	977	980	984	987	990	994	997	1001	1004	1007	1011	1014	1018	1021	1024	1028	1031	1035
54	965	968	972	975	978	982	985	988	992	995	999	1002	1005	1009	1012	1016	1019	1022	1026	1029	1032
55	963	966	970	973	977	980	983	987	990	993	997	1000	1004	1007	1010	1014	1017	1020	1024	1027	1031
56	961	964	968	971	975	978	981	985	988	991	995	998	1001	1005	1008	1012	1015	1018	1022	1025	1028
57	959	963	966	969	973	976	980	983	986	990	993	996	1000	1003	1006	1010	1013	1016	1020	1023	1027
58	957	961	964	967	971	974	977	981	984	988	991	994	998	1001	1004	1008	1011	1014	1018	1021	1025
59	956	959	962	966	969	972	976	979	982	986	989	992	996	999	1003	1006	1009	1013	1016	1019	1023
60	954	957	960	964	967	970	974	977	980	984	987	990	994	997	1000	1004	1007	1010	1014	1017	1021
61	952	955	959	962	965	969	972	975	979	982	985	989	992	995	999	1002	1005	1009	1012	1015	1019
62	950	953	957	960	963	967	970	973	977	980	983	987	990	993	997	....	1003	1007	1010	1013	1017
63	948	952	955	958	962	965	968	971	975	978	981	985	988	991	995	998	1001	1005	1008	1011	1015
64	946	950	953	956	960	963	966	970	973	976	980	983	986	990	993	996	1000	1003	1006	1009	1013
65	944	948	951	954	958	961	964	968	971	974	978	981	984	988	991	994	998	1001	1004	1007	1011
66	943	946	949	953	956	959	963	966	969	973	976	979	982	986	989	992	996	999	1002	1006	1009
67	941	944	948	951	954	958	961	964	967	971	974	977	981	984	987	991	994	997	1001	1004	1007
68	939	942	946	949	952	956	959	962	965	969	972	975	979	982	985	989	992	995	998	1002	1005
69	937	941	944	947	951	954	957	960	964	967	970	974	977	980	983	987	990	993	997	1000	1003
70	936	939	942	946	949	952	955	959	962	965	968	972	975	978	982	985	988	991	995	998	1001
71	934	937	940	944	947	950	954	957	960	963	967	970	973	977	980	983	986	990	993	996	999
72	932	936	939	942	945	949	952	955	958	962	965	968	971	975	978	981	985	988	991	994	998
73	930	933	937	940	943	947	950	953	956	960	963	966	969	973	976	979	982	986	989	992	996
74	929	932	935	938	942	945	948	951	955	958	961	964	968	971	974	977	981	984	987	990	994
75	927	930	933	937	940	943	946	950	953	956	959	963	966	969	972	976	979	982	985	989	992
76	925	928	932	935	938	941	945	948	951	954	958	961	964	967	971	974	977	980	984	987	990
77	923	927	930	933	936	940	943	946	949	953	956	959	962	966	969	972	975	978	982	985	988
78	922	925	928	931	935	938	941	944	948	951	954	957	960	964	967	970	973	977	980	983	986
79	920	923	926	929	933	936	939	943	946	949	952	955	959	962	965	968	972	975	978	981	985
80	918	921	925	928	931	934	938	941	944	947	950	954	957	960	963	967	970	973	976	979	983
81	917	920	923	926	929	933	936	939	942	945	949	952	955	958	962	965	968	971	974	978	981
82	915	918	921	924	928	931	934	937	941	944	947	950	953	957	960	963	966	969	973	976	979
83	913	916	920	923	926	929	932	936	939	942	945	948	952	955	958	961	964	968	971	974	977
84	911	915	918	921	924	927	931	934	937	940	943	947	950	953	956	959	963	966	969	972	975
85	910	913	916	919	922	926	929	932	935	938	942	945	948	951	954	958	961	964	967	970	974
86	908	911	915	918	921	924	927	930	934	937	940	943	946	950	953	956	959	962	965	969	972
87	907	910	912	916	919	922	926	929	931	935	938	942	945	948	951	954	957	961	964	967	970
88	905	908	911	914	918	921	924	927	930	933	937	940	943	946	949	952	956	959	962	965	968
89	903	906	909	913	916	919	922	925	929	932	935	938	941	944	948	951	954	957	960	963	967
90	901	905	908	911	914	917	920	924	927	930	933	936	939	943	946	949	952	955	958	962	965
91	900	903	906	909	913	916	919	922	925	928	932	935	938	941	944	947	951	954	957	960	963
92	898	901	905	908	911	914	917	920	924	927	930	933	936	939	942	946	949	952	955	958	961
93	897	900	903	906	909	912	915	919	922	925	928	931	934	937	941	944	947	950	953	956	960
94	895	898	901	904	907	911	914	917	920	923	926	929	933	936	939	942	945	948	951	955	958
95	893	897	900	903	906	909	912	915	919	922	925	928	931	934	937	940	944	947	950	953	956
96	892	895	898	901	904	907	911	914	917	920	923	926	929	932	936	939	942	945	948	951	954
97	890	893	896	900	903	906	909	912	915	918	921	925	928	931	934	937	940	943	946	949	953
98	889	892	895	898	901	904	907	910	914	917	920	923	926	929	932	935	939	942	945	948	951
99	887	890	893	896	899	902	906	909	912	915	918	921	924	927	930	934	937	940	943	946	949
100	885	889	892	895	898	901	904	907	910	913	917	920	923	926	929	932	935	938	941	945	948

an insufficient amount of oxygen remains after the first explosion. In such case the gas is drawn back into the burette and the oxygen and nitrogen mixture is added until the volume approaches 100 c. c.; the gases are again mixed, exploded and the analysis completed as usual, the oxygen added being found from the sum of the two additions. If, in the first explosion of any analysis, the ratio of inert to combining gases is not sufficiently high, and doubt exists as to the formation of oxides of nitrogen, the explosion and resulting determinations may be duplicated, using a larger amount of oxygen and nitrogen mixture and the portion of the residue which has been reserved in the phosphorus pipette, or the residue from a new portion obtained in the following manner. One hundred cubic centimeters are taken from the original sample and the carbon dioxide, oxygen, and illuminants if present, are removed in the usual manner. The remaining residue will not be the same as that obtained on the first sample on account of the reduction in volume of the carbonic acid, due to its solubility in the water in the sample tube, but a part of the residue is taken equal in volume to the gas used in the first explosion, and the analysis is continued as usual.

3. **Producer Gas.**—For the analysis of producer gas the method as described under Blast Furnace Gas is used with one or two modifications as subsequently described.

Determination of Carbon Dioxide and Illuminants. A sample of exactly 100 c. c. is measured and the carbon dioxide removed in the usual manner. Next the gas is bubbled twice through the fuming sulphuric acid pipette to remove

the unsaturated hydrocarbons, composed chiefly of ethylene and commonly called illuminants. Care must be exercised not to allow any water to enter the pipette. After the acid has been drawn to its initial point in the stem of the pipette, the gas is passed into the solution of potassium hydrate two or three times to remove the white acid fumes. A reading is taken and the decrease in volume noted indicates the percentage of illuminants present.

Determination of Oxygen. Next the oxygen is removed, and since the presence of illuminants entirely or partially inhibits the desired reaction with stick phosphorus, it is essential that they be removed before this is attempted.

Determination of Carbon Monoxide, Hydrogen and Methane. Producer gas is much richer in combustible gases than blast furnace gas; therefore the extent of dilution of the residue taken for the explosion should be such as to reduce to the minimum the temperature of the latter, and thereby the error due to formation of oxides of nitrogen. It is also desired that the maximum amount of the original residue practicable enter the explosion, in order that multiplying errors due to fractionation of the sample may be reduced to the minimum. The method to be used for the determination of these constituents is dependent upon the percentage of combustible gases in the sample. This percentage usually varies inversely with that of the carbon dioxide present.

Method "A" is used when less than seven per cent of carbon dioxide is present in the sample. One-half of the residue remaining after the oxygen content has been determined is rejected or reserved in the phosphorus pipette. The remaining half is passed into the explosion pipette, the stem of the latter being completely filled with water. Approximately 70 c. c. of oxygen and nitrogen mixture are added, the mixture in the intervening capillary being swept into the burette by the water remaining in the stem of the explosion pipette. Then the oxygen and nitrogen mixture is accurately measured, mixed with the above residue and exploded; the gases are returned to the burette, allowed to cool and a reading is taken. Contraction I is found by subtracting the residue left after the explosion from the sum of the residue taken for the explosion and the auxiliary gas added. Contraction II, the amount of carbonic acid formed during the explosion, is next determined in the regular manner and the residue is passed into the explosion pipette. Approximately 70 c. c. of hydrogen are added, as just described with the oxygen and nitrogen mixture, and accurately measured, mixed with the above residue and exploded, the resulting gases being returned to the burette and measured in the usual way. Contraction III is found in the same manner as was Contraction I.

The expected amount of combustibles may not be present in the gas portion taken for the explosion, and the residue after the first or second explosion may be greater than can be measured in the burette. In case this should occur, the residue is measured in two portions, the sum of which plus 0.1 cc, the capillary space, equals the total residue.

Example—Data.

Burette Readings.

	0.0	
Potassium hydrate		4.3 Carbon dioxide.
	4.3	
Fuming sulphuric		.6 Illuminants.
	4.9	
Stick phosphorus		0.0 Oxygen.
	4.9	
		95.1 Residue.
		47.5 ½ Residue.
		52.5 Water level.
Pass residue into explosion pipette		
	100.0	
Add oxygen and nitrogen mixture		70.4 Mixture added, 45% oxygen.
	29.6	31.7 Oxygen added.
Explosion	0.8	99.2 Residue.
		47.5 + 70.4 − 99.2 = 18.7 Contraction I.
Potassium hydrate		14.2 Contraction II.
	15.0	85.0 Residue.
Pass residue into explosion pipette		
	100.0	
Add hydrogen		70.3 Hydrogen added.
	29.7	
Explosion	2.7	97.3 Residue.
		85.0 + 70.3 − 97.3 = 58.0 Contraction III.

Calculations.

31.7	Oxygen added.
19.3	Oxygen in excess, or ⅓ Contraction III.
12.4	Oxygen consumed in explosion.
18.7	Contraction I.
6.3	Hydrogen present.
18.7	Contraction I.
56.8	4 × Contraction II.
58.0	Contraction III.
133.5	Total.
44.5	⅓ Total.
31.7	Oxygen added.
12.8	Carbon monoxide present.
14.2	
1.4	Methane present.

Analysis.	Per cent.
Carbon dioxide	4.3
Illuminants	0.6
Oxygen	0.0
Carbon monoxide	25.6
Methane	2.8
Hydrogen	12.6
Nitrogen	54.1
	100.0

Explosion Ratios: I, 2.58; II, 1.68.

Method "B" is used when the carbon dioxide of the sample is above 7 per cent. The method of analysis calculation is identical with those described under Blast Furnace Gas, with the exception that oxygen and nitrogen mixture is added to the portion of the residue reserved for the explosion until the volume approaches 100 cc. If the carbon dioxide value is between 8.0 and 9.0 per cent the amount of oxygen and nitrogen mixture to be added should be diminished by about 10 cc or difficulty may be encountered in removing the excess oxygen in the second explosion.

Example—Data.

Burette Readings.

	0.0	
Potassium hydrate		7.5 Carbon dioxide.
	7.5	
Fuming Sulphuric		.0.1 Illuminants.
	7.6	
Stick phosphorus		0.0 Oxygen.
	7.6	
		92.4 Residue.
		46.2 ½ Residue.
		53.8 Water level.

Add oxygen and nitrogen mixture	52.5 Mixture added, 44.5% oxygen.
1.3	23.4 Oxygen added.
Explosion	16.2 Contraction I.
17.5	
Potassium hydrate	11.2 Contraction II.
28.7	
Add hydrogen	28.7 Hydrogen added.
0.0	
Explosion	39.0 Contraction III.
39.0	
	13.0 ⅓ Contraction III.

Calculations.

23.4 Oxygen added.
13.0 Oxygen in excess or ⅓ Contraction III.

10.4 Oxygen consumed in explosion.
16.2 Contraction I.

5.8 Hydrogen present.

16.2 Contraction I.
44.8 4 × Contraction II.
39.0 Contraction III.

100.0 Total.

33.3 ⅓ Total.
23.4 Oxygen added.

9.9 Carbon Monoxide present.
11.2 Contraction II.

1.3 Methane present.

Analysis.	Per cent.
Carbon dioxide.....	7.5
Illuminants	.1
Oxygen	.0
Carbon Monoxide	19.8
Methane	2.6
Hydrogen	11.6
Nitrogen	58.4
	100.0

Explosion Ratios: I, 2.60; II, 1.56.

4. **Flue Gas.**—The repeated determination of large quantities of oxygen by stick phosphorus or pyrogallol solution introduces undesirable features with the use of either reagent. In the former case, unless care is exercised, the heat of reaction causes fusion, with consequent disarrangement of the phosphorus sticks; with the use of the latter reagent there is a liability of the incomplete removal of all the oxygen. In such a case it would be absorbed in the cuprous chloride solution with the carbon monoxide, if this reagent were used for the purpose, thus giving erroneous values to the carbon monoxide found. To overcome these difficulties the following method has been devised whereby the oxygen and carbon monoxide are determined by explosion, and it has been proven superior to the absorption method, which is the method usually followed in the analysis of flue gases.

Determination of Carbon Dioxide. This constituent is determined on exactly 100 cc, measured in the usual way, as under blast furnace gas.

Determination of Oxygen and Carbon Monoxide. One-half the volume of the residual gases is computed and rejected. The oxygen present in the sample varies inversely with the carbon dioxide content, consequently the hydrogen to be added for its combustion is dependent upon this factor. If the sample contains less than 10 per cent of carbon dioxide, approximately 20 cc of hydrogen and 30 cc of oxygen and nitrogen mixture are added to the reserved portion of the residue. If the sample contains more than 10 per cent of carbon dioxide, 15 cc of hydrogen and 35 cc of oxygen and nitrogen mixture are added The gases are thoroughly mixed, exploded, and after the proper interval Contraction I is noted. They are then passed through the solution of potassium hydrate to remove the carbon dioxide that has been formed from the burning of the carbon monoxide that may have been present, and Contraction II is noted.

To the residue hydrogen is added until the volume of the mixture approaches 100 cc. The gases are thoroughly mixed and exploded, drawn back into the burrette, and after the proper interval a reading is taken and Contraction III is noted.

The data necessary for the calculation of the percentage of oxygen and carbon monoxide in the sample has thus been obtained. From the reactions given under the determination of carbon monoxide, hydrogen and methane in blast furnace gas it is seen that when one volume of oxygen unites with two volumes carbon monoxide there is a contraction of one volume and the formation of two volumes of carbon dioxide, which on absorption cause a further contraction of two volumes; the total contraction is, therefore, three volumes for one volume of oxygen, or the oxygen is one-third the total contraction. The carbon monoxide was also shown to have the same volume as the dioxide formed. In the combustion of hydrogen and oxygen it was shown that the oxygen is one-third the contraction. Then the oxygen entering the two explosions, or the total oxygen, is one-third the sum of the three contractions and the oxygen in the sample is found by subtracting the oxygen added from this total oxygen. These facts are expressed in the following equations:

$$\text{Carbon monoxide} = \text{Contraction II.}$$

$$\text{Oxygen in sample} = \frac{\text{Con. I} + \text{Con. II} + \text{Con. III}}{3} - \text{Oxygen added.}$$

Burette Readings.

0.0	
Sodium hydrate	9.2% Carbon Dioxide.
9.2	
90.8	Residue.
45.4	½ Residue.
54.6	Water level.
Add hydrogen	19.8 Hydrogen added.
34.8	
Add oxygen and nitrogen mixture.....	29.9 Oxygen and nitrogen mixture added, 44.4% oxygen.
	13.3 oxygen added.
4.9	
Explosion	29.7 Contraction I.
34.6	
Sodium hydrate	.0 Contraction II.
34.6	
Add hydrogen	34.4 Hydrogen added.
0.2	
Explosion	25.2 Contraction III.
25.4	

Calculations.

0.0 Contraction II, or Carbon monoxide present.

29.7 Contraction I.
0.0 Contraction II.
25.2 Contraction III.

54.9 Total.

18.3 ⅓ total, or total oxygen.
13.3 Oxygen added.

5.0 Oxygen present.

Analysis.	Per cent.
Carbon dioxide.....	9.2
Oxygen	10.0
Carbon monoxide.....	0.0
Nitrogen	80.8
	100.0

Explosive ratios. I, 2.20; II, 2.96.

The absence of methane and hydrogen in a flue gas may be proven by the application of this formula:

$$H_2+CH_4=\frac{2(\text{Con. I})-\text{Con. II}}{3}-H_2 \text{ added for the first explosion.}$$

This is usually found to be zero.

Thermal Values.—If a calorimeter is used for determining the thermal value of a gas, the Junkers instrument is recommended for this purpose. The results so obtained must be expressed in b.t.u. under the standard conditions of dry gas at 62° Fahr. and 30 in. mercury.

When a calorimeter is not available it is customary to calculate the calorific value of a gas from the analytical results. Considerable discrepancies exist among the values commonly given as heats of combustion for the different gases; but it is believed that the following figures, being means of published values of several well-known investigators, will give results which are sufficiently reliable for all technical purposes.

In the following table, showing the b.t.u. developed in the combustion of 1 cu. ft. of the lifferent gases, measured at 62° Fahr. and 30 in. mercury, two values are given—the gross and the net, the difference between them being the latent heat of the water formed, which is not included in the latter. The products of combustion in both cases are assumed to be at 62° Fahr. For the purpose of comparison all Corporation results should be expressed in terms of net b.t.u.

	Gross.	Net.
Ethylene (illuminants)	1591	1491
Carbon monoxide	324	324
Methane	1012	912
Hydrogen	328	278

In the calculation of the thermal value of a gas the percentage of each combustible present is multiplied by its value in the above table, and the sum of the products will represent the b.t.u. evolved in the combustion of 1 cu. ft. of the gas.

Specific Gravity and Percentage by Weight.—When suitable apparatus is not available for determining the specific gravity of gas, it may be calculated from the volumetric analysis by means of the following table compiled by the Bureau of Standards, showing the specific gravity of various gases compared with air:

Air	1.0000
Carbon dioxide	1.5290
Ethylene (illuminants)	.9684
Oxygen	1.1054
Carbon monoxide	.9670
Methane	.5545
Hydrogen	.06951
Nitrogen	.9674
Water vapor	.6222

The calculated specific gravity of a gas is found by adding the products obtained by multiplying the percentage by volume of each constituent gas by its specific gravity. The respective products thus obtained divided by the specific gravity of the gas yield the percentages by weight of the constituent gases.

The specific gravity of moist gas is found by adding the product of the specific gravity and the percentage of water vapor accompanying dry gas to the specific gravity of the dry gas, and dividing the sum by one plus the percentage of water vapor. This latter value is found by multiplying the grains per cubic foot dry gas by .3012, the volume occupied by 1 grain water vapor, or by dividing the vapor pressure by the difference of the gas pressure and the vapor pressure.

Weight of a Cubic Foot of Gas.—The following table, calculated from the most recent values given by the Bureau of Standards, shows the weight in pounds of 1 cu. ft. of the constituent gases at 62° and 30-in. pressure. The weight of 1 cu. ft. of gas is the sum of the values found by multiplying the weight per cubic foot of each of the constituent gases by its percentage by volume.

Air	.07626
Carbon dioxide	.11660
Ethylene	.07386
Oxygen	.08430
Carbon monoxide	.07375
Methane	.04228
Hydrogen	.005302
Nitrogen	.07377
Water vapor	.04745

The weight of 1 cu. ft. of moist gas is found by adding the product of the weight of a cu. ft. of water vapor and the percentage of water vapor accompanying dry gas to the weight of dry gas, and dividing the sum by one plus the percentage of water vapor.

Correction for Temperature and Pressure.—All metered gas shall be corrected to dry gas at 62° Fahr. and 30 in. of mercury pressure as the standard conditions for comparison.

When T = Temperature of gas at meter plus 459°.

T_s = Temperature sought plus 459°.

P = Pressure found by barometer + manometer reading at meter — vapor pressure at temperature of gas, and

P_s = Pressure sought.

Then a factor may be found by calculation, or from the table on page 358, equal to $\frac{T_s P}{P_s T}$ and applied as follows: The volume at standard conditions equals the volume observed times the factor. The b.t.u. at standard conditions equal the b.t.u. observed divided by the factor.

Vanadium in Cast Iron.*

BY GEORGE L. NORRIS.

The use of vanadium in gray iron castings has been limited when compared to its use in steel castings, although the past year has shown a marked increase and its use in certain classes of castings has become well established. We have in the case of cast iron a material the cost of producing which from the crude pig iron has been slight. The price at which iron castings in general are sold precludes the use of any alloy for affecting the quality of the material, except in the case of certain classes of castings where unusual strength or some special quality in the iron is desired. In such cases even a very material increase in the cost of the castings is readily accepted.

In the case of vanadium we are dealing with a metal of fairly high price and its use in the manufacture of cast iron naturally will be confined to the classes of work that can stand a material increase in cost in return for the special qualities obtained. The addition of 1/10 per cent of vanadium to cast iron at the present price of $5 per pound means an increase of ½ cent a pound for the iron in the ladle and necessitates an increase in selling price of ¾ cent to 1 cent a pound at least to cover losses due to scrap, gates, spruces, etc.

While vanadium has a strong affinity for oxygen and nitrogen and is partly used up by combining with these elements, the primary purpose of adding vanadium to cast iron is not the removal of oxygen and nitrogen. The removal of oxygen can be effected sufficiently well by cheaper means and as the injurious effect of small amounts of nitrogen on a refined metal, such as steel, is still an open question, the harmfulness of this element on a metallurgically crude metal like cast iron is open to still greater doubt.

Vanadium increases the strength of the cast iron to which it is added from 10 per cent to 25 per cent, depending, of course, upon the initial strength of the iron. This increase of strength is due directly to the intensifying effect of the vana-

*Abstract of a paper presented before the American Foundrymen's Association at the Pittsburgh Convention.

dium on the metals present in the cast-iron alloy, such as silicon and manganese and also by combination with the chemically combined carbon.

It exercises a very strong effect on the grain of the iron, causing a more even distribution of the graphite with consequent greater freedom from both hard and porous spots. This effect is very apparent, vanadium cast-iron fractures showing a finer and more uniform grain. In the case of chilled iron, vanadium produces a deeper, stronger and tougher chill, and one less liable to spall or flake. This chill, however, is not as hard apparently as the more brittle chill of ordinary chilling iron, and can be filed and machined more readily than the latter.

Vanadium also gives an element of toughness to cast iron which is very apparent in machining the castings. This toughening effect, together with the uniform fineness of grain or texture due to even distribution of graphite in small flakes, makes vanadium cast iron very superior for such purposes as gas and steam-engine cylinders, glass-bottle molds, gears, castings to withstand heat and internal pressures of fluids and gases without leakage.

Three or four years ago the New York Central lines equipped one of its locomotives with a pair of vanadium cast-iron cylinders. The performance of this pair of cylinders over a test period of about two years was so satisfactory that 500 or more locomotives on these lines are now equipped with vanadium cylinders. The test of the original pair demonstrated that over twice the normal mileage of ordinary cast-iron cylinders could be obtained from vanadium cylinders before reboring. The loss in casting these cylinders was extremely low, no cylinders being scrapped at all for the first 183 locomotives ordered.

The same sort of results have been reported by a builder of large gasoline engines for marine use. In his particular case the number of cylinders formerly lost when tested after machining was excessive, due to porousness of certain sections.

Piston rings and cylinder and valve bushings of vanadium cast iron have been thoroughly tried out and adopted by a number of leading railroads on account of the greatly increased length of service obtained. A very interesting application of vanadium has been for glass-bottle molds. These are chill castings and are usually covered with engraving. The iron has to be clean, close and uniform in grain, machineable and take a good polish. The first vanadium cast-iron bottle molds were made a little over a year ago and proved not only to have a longer life, but it was found possible to increase the speed of the bottle machines as the glass did not appear to stick to these molds to any extent. This can be readily accounted for by the uniformity of the grain and freedom from mottled spots.

One of the first applications of vanadium to cast iron was in chilled iron rolls for sheet mills. A number of these were made and gave excellent results, some of them showing 50 per cent greater tonnage with one-half of the dressing against the regular chilled roll. The question of cost, however, entered in here in an unusual manner. Fully 65 per cent to 70 per cent of the roll breakages are due to accidental causes and no amount of vanadium would insure, for instance, against a roller letting a pair of tongs pass through the rolls.

We have had opportunity from time to time to make comparative tests of the strength of cast iron with and without vanadium extending over a considerable period, and believe that these comparisons are more nearly representative than single tests, as they represent average conditions.

The following tests are the average for ten consecutive days' work on locomotive cylinders. The transverse tests were made on bars 1 in. square, 12 in. between supports; the bars were machined all over and consequently were absolutely comparable, which is not always the case with bars only brushed or cleaned.

	Transverse.	Tensile.
Plain cast iron	2130 lb.	24,225 lb.
Vanadium cast iron ..	2318 "	28,728 "

This iron has stood 750-lb. water pressure for sections ¾ in. thick. The average chemical analysis of this iron is about as follows:

Combined carbon	0.50
Silicon	1.50
Manganese	0.45
Phosphorous	0.600
Sulphur	0.095

I endeavored to get together a series of tests on irons of various compositions to show the effect of vanadium both on the strength and on the chilling qualities, but owing to various delays I am not able to present this in complete form. The following tests were supposed to be from iron from the same ladle, before and after adding vanadium, but the chemical analysis shows that such could not have been the case. The difference in composition, however, so far as it influences the physical qualities of the iron are all in favor of the iron without vanadium. In addition to the bars for tensile test, chill blocks were cast to show the effect of vanadium on the depth of the chill.

CHEMICAL COMPOSITION.

	Without Vanadium.	With Vanadium.
Graphitic carbon	2.34	2.48
Combined carbon	0.79	0.71
Silicon	1.83	2.21
Manganese	0.43	0.424
Phosphorus	0.508	0.546
Sulphur	0.102	0.071
Vanadium	.....	0.056

PHYSICAL TESTS.

	Without Vanadium.	With Vanadium.
Tensile strength	34,500	35,600
Depth of chill.....	No white chill	1/16 in. white chill

Size of chill test block 4½ in. square by 1½ in. thick.

The difference in the machining quality of the two irons was decidedly in favor of the vanadium iron. I need hardly mention that the cast iron treated with vanadium should be at least approximately of the proper composition for the particular class of work to which it is intended.

The usual amount of vanadium added to cast iron is 1/10 per cent for cupola melted iron, equivalent to about 5 ounces of ferro-vanadium for each 100 lb. Some foundries advocate the use of a higher percentage, up to 15 per cent, which is about as much as can be conveniently added in the ladle. As low a percentage as 0.05 per cent will produce distinctive effects.

Owing to the high melting point of vanadium and the limited reserve heat available in a ladle of iron it is advisable to use the alloy in a finely crushed condition, or even powdered, when the amount of iron treated is small, and to use an alloy of low melting point. The iron should be tapped out as hot as possible and a ladle used that has just been emptied in order to conserve as much heat as possible.

After the bottom of the ladle is covered with a few inches of iron the finely crushed or powdered ferro-vanadium is added by sprinkling on the stream as it flows down the spout to the ladle; in this way advantage is taken of all the available heat and also of the mixing effect of the stream as it strikes the iron in the ladle. After the vanadium is added the contents of the ladle should be well stirred and allowed to stand a few moments in order to insure thorough incorporation and complete reaction.

In the case of the higher-grade air-furnace iron, with its reserve of available furnace heat, the procedure is very simple. After the charge is melted and 15 to 20 minutes before tapping the ferrovanadium is added and the bath well stirred or

rabbled. The addition of 0.18 per cent to 0.20 per cent vanadium is recommended in this case, equivalent to 10 ounces to 11 ounces of 35 per cent ferrovanadium per 100 lb. of metal.

The American Vanadium Company has lately developed a ferrovanadium containing about 30 per cent to 35 per cent vanadium, 10 per cent to 15 per cent silicon, 5 per cent to 10 per cent manganese and 2 per cent to 5 per cent aluminium, which melts and dissolves readily in molten cast iron and has been giving very satisfactory results. In distinction from standard low silicon "Amervan" ferrovanadium this new alloy is called "Masvan."

Melting Non-Ferrous Metals in an Electric Furnace.

BY CHAS. A. WEEKS, M. E.

In an address by Dr. J. W. Richards, of Lehigh University, at the convention of the American Brass Foundrymen's Association, held at Detroit, June, 1910, the possibilities of melting bronze and brass in an electric furnace were compared in a theoretical way with the present methods of melting these metals in oil-direct-fired and crucible coke or oil-heated furnaces.

For several years I have been investigating the use of an electric furnace for melting copper, monel metal, bronze, brass and zinc refining, in large units and on a commercial scale, comparing the costs of melting in my electric furnace with other methods of melting now in use, along the lines of Dr. Richards' address.

In order that this work be done in a large and original way, I designed and built an electric furnace with a crucible capacity of three tons of copper. My knowledge of non-ferrous metals melting caused me to take rather radical steps to prevent the copper and other volatilizable metal from being volatized as they would be in the general arc type of furnace. To prevent this burning of metal I designed a furnace which may be classified under the arc type, but in which the heat is transferred to the metal by radiation from the arc, together with some reflection and conduction from walls and roof of furnace. The furnace is entirely enclosed and the atmosphere is neutral, the furnace works very smoothly, heating any size of charge up to furnace capacity; it is built economically and with no lining complications, it is as simple as an open-hearth furnace.

The furnace used in making these runs was also provided with a condensing system for the treatment and recovery of zinc from zinc dross and zinc ores containing copper, iron, lead, etc. While unfortunately I was not able to treat ores at the time of running the furnace during July and August, 1910, yet at that time I made a very good zinc from impure zinc dross, and during the run made some 1600 lb. of zinc and no blue powder. Some zinc oxide was produced at vent of condenser, but this was wanted and expected. It is my belief that this is the largest amount of zinc produced in an electric furnace, without making blue powder, made at a commercial cost. The run was stopped at that time by reason of making more zinc than the condenser could handle.

Copper Melting.

A number of runs were made melting copper scrap in which charges of about 2000 lb. were melted. To melt and superheat to a temperature above 1600° C. from cold furnace required 600 kw-hours, and the metal was poured in 3½ hours. This temperature was considerably in excess of copper melting, using more power and longer time. The loss of copper with new lining was less than 1 per cent.

Melting another charge of 2000 lb. to temperature of 1200° C. required from cold furnace about 500 kw-hours, time three hours. Under operating conditions, that is, with a hot furnace and using charges of two tons or more, this figure would undoubtedly be cut down to 300 kw-hours or less per ton, metal heated to pouring temperature of 1200° C., with melting time about 2½ hours.

As no slag complications are involved, the furnace can be tapped into as many receptacles as wanted without power interruption; the copper can be poled at will and resulting tough pitch copper can be rested in the furnace indefinitely without becoming oxidized, since a blanket of charcoal may be kept floating on the metal to protect it from such slight amounts of air as do enter a practically tight furnace.

This furnace is entirely free from the very objectionable vaporization of copper as experienced with those furnaces in which the current arcs from the electrodes to the bath, with resultant danger to the operator, as noticed in the letter of Mr. C. A. Hansen in the February, 1911, number of this journal.

The above figures will apply to melting monel metal, the proper pouring temperature of which is around 1400° C. The melting of monel metal in an open-hearth or reverberatory furnace or in crucibles allows a very serious oxidation of the metal; as a general result the castings made are porous for high-pressure work. It is only necessary to say that melting this metal in a non-oxidizing furnace, such as my electric furnace, where the metal cannot be burnt or oxidized, the making ing large sound castings from this metal will be accomplished.

Bronze and Brass Melting.

In his address Dr. Richards gave 1 lb. of coke = 0.67 lb. of oil = 1.8 hp-hours or coke at $4 per ton = oil at $6 = power at $12.25 per hp-year. I have taken power at 1 cent per kw-hour and would say that while this cost seems low a well-known oil-engine concern will guarantee a cost of about ½ cent per hp-hour. Any large electric plant can sell power at this price at a profit for day load. In order to obtain some idea of the causes controlling oxidation and cost comparative runs on gas, coke and oil-fired furnaces show that oxidation varies approximately as the length of time required to melt; also in proportion to the square of the temperature. On this basis, comparing the electric furnace with direct oil-fired and coke-fired crucible furnaces, I have assumed for the former an oxidation of 0.7 per cent (actually 0.5 per cent should be taken).

Taking the amount of power necessary to melt one ton of bronze or brass at 150 kw-hours, I find a reasonable figure to be 190 kw-hours per ton in 1½ hours time.

For comparison with the above figures I will take figures given in a test of rival direct-heating oil furnaces. This test was made under the direct supervision of one of the best-known brass foundry superintendents, who is in charge of one of the largest foundries in the country.

This test was fair and the figures are presumably correct. The metal mixture was 66 parts of copper, zinc 34 parts, part scrap, gates, etc. Pouring temperature, 2135° F.; loss of metal through oxidation, 1 1/3 per cent; fuel, 36 gal. of oil per ton metal melted, at 3 cents per gal.; cost of lining, taken at 10 cents per ton.

Coke-fired furnace; coke $4 per ton; melting 8 lb. of metal to 1 lb. of coke, with oxidation 1 per cent; cost of crucibles $3 per ton metal melted, with copper at 13 cents per lb., zinc 5 cents per lb.

We thus get the following:

	Electric furnace.	Direct oil-fired furnace.	Coke-heated crucible furnace.
Cost of heat....	$1.90	$1.08*	$0.50*
Oxidation	1.44	2.68	2.05
Linings	0.10	0.10	0.10
Crucible	0.00	0.00	3.00
	$3.44	$3.86	$5.65
Time of melting	1½ hours	2 to 2½ hours	2 to 3 hours

While the electric furnace shows a lower cost and larger output, as above, melting in two-ton charges, the power cost

*Furnace builders' claim.

would be cut to 150 kw-hours or $1.50 per ton metal melted in 1½ hours.

Interest, depreciation or labor are not taken into consideration in above.

As is well known, copper, monel metal, bronze or brass cannot be melted in any type of fuel-fired furnace without oxidation, as the average crucible at red heat is metal-tight, but not gas-tight, gases readily finding their way through the pores of the crucible material. We also know that carbon dioxide gives up oxygen very readily to many metals, especially to zinc, being reduced to carbon monoxide, so that the fact of my electric furnace being sealed and non-oxidizing, makes it particularly valuable in melting non-ferrous metals.

The best furnace is the one that produces the best metal, from which the greater number of perfect castings can be made, in which the temperature is easily controlled, and in which a neutral atmosphere can be maintained for any length of time at required temperature without oxidizing and little loss of metal, in which metal can be treated or alloyed with large output without the production of deleterious fumes and with perfect safety to operator, I believe there can be no question as to the superiority of my arc-heat radiating furnace for the melting of non-ferrous metals in these respects. Compared with the direct oil-fired, coke or oil-heated crucible furnaces, it certainly makes better metal at a cost much less than was thought possible commercially. I here quote a paper of Dr. Richards and discussion as to actual present cost of melting yellow brass.

Efficiencies of Furnaces.

In a paper presented by Dr. J. W. Richards at the Pittsburgh meeting of the American Brass Founders' Association in May, 1911, the author explained what was meant by efficiency of industrial appliances, and cited examples of carelessness in describing the efficiencies of various products. Applied to furnaces, efficiency often means ratio of fuel to output.

Mr. Webster asked for information relative to the melting ratio of brass furnaces and stated that in using anthracite coal the usual ratio was from 2½ to 3 lb. of metal melted per pound of fuel. The question was then asked how this compared with regular practice, and Mr. W. M. Corse stated that with coke the average is 45 lb. per 100 lb. of metal = 2.25 lb. of metal per pound of coke.

Mr. H. P. MacKinnon, of the Hersey Manufacturing Company, South Boston, Mass., stated that he had melted 900,000 lb. of bronze for light castings, at an average consumption of 3¾ gal. of oil per 100 lb. of metal in a tilting furnace, and that approximately one-ninth of the metal was in form of borings. The cost of oil averaged 18 cents for 4½ gal., 75 gal. of oil melted 2000 lb. of metal at cost of 4 cents per gallon.

Mr. John A. Smith, of the American Air Brake Company, stated his fuel consumption was 3 gal. of oil per 100 lb. of metal in an open-flame furnace or 60 gal. per ton.

Fuel cost of melting 2000 lb. brass as above:

Coke, 900 lb., at $4 per ton	$1.80
Oil tilting crucible furnace, 75 gal., at 4 cents	3.00
Oil, open-flame furnace, 60 gal., at 4 cents	2.40
C. A. Weeks electric furnace, 1 cent per kw-hour, 190 or 150 kw-hours	$1.90 or $1.50

Temperature of yellow brass ready for pouring, 2000° F.

Refining of Zinc from Impure Zinc Dross in an Electric Furnace.

Some years ago several plants were erected, using methods and furnaces from my designs, for the work of recovering a pure zinc from impure zinc dross, purchased in open market, without selection.

The requirements of these plants were that the zinc recovered should be equal to the well-known horse-head brand of zinc in every way. The cost of refining to be low, production regular and furnaces of long life. I am glad to say that these conditions were met in every way, particularly as to furnace upkeep. The repair account in some five years has been less than 1 per cent of the first cost. As these furnaces have been under fire with heat high enough to volatize zinc, through retorts for a year at a time, they show fair construction. While the above is not under discussion I state it to show that I have some knowledge of this subject.

The greatest cost and uncertainty of the above process are the retorts, which are of small capacity, the largest size of retort holding 1000 lb. The cost of retorts per ton of dross charged is from $5 to $8 and in some cases where retorts are not properly treated the cost is greatly increased.

With a view of obtaining, if possible, a better product with larger charges, and in some cases to use a more impure dross and do away with retorts, I was led to design an electric furnace for this work, in which I could charge two to three tons, with an output of not less than 200 lb. of refined zinc per hour.

When I took up the construction of this furnace, which is of the arc type, from which heat is transferred to the charge by radiation from the arc, with some reflection and conduction from the furnace walls and roof, at that time (1905-6) the statement was made that all attempts to refine zinc metal from either zinc ores or zinc dross in presence of an electric arc had been failures, as the vapors were so superheated by the intense heat of the arc that it had been impossible to condense them; further, that when the zinc vapors did condense they produced blue powder. In this relation I would say that from the very first attempts I made to refine zinc dross, some ten years ago, I have never made during this entire period more than 5 lb. of blue powder in actual working.

As I was not particularly interested in blue powder and did want zinc metal, I was not interested in the failure of others. I built the furnace for this work, arranged to use a condenser, which I had having a capacity of 35 to 50 lb. metal per hour, intending to build a larger condenser later, with capacity of 300 lb. metal per hour.

After preliminary heating to dry out the lining and testing the working of furnace, which was found to operate very smoothly, we made several runs on zinc dross, and in the last run obtained over 1000 lb. of zinc metal. But owing to insufficient condensing capacity with an average output of 35 lb. per hour, and zinc being made so fast that metallic zinc jammed everything up, the run was stopped.

During the run no blue powder was made, liquid zinc coming off regularly for some 36 hours; during this time there was no adjustment or appreciable loss of electrodes.

To give some idea of the furnace working, I would say that with 80 kw input 35 lb. of zinc was obtained per hour; of this, by reason of the excessive overheating of the furnace, due to the small amount of zinc taken off, about 65 kw was lost by radiation and water cooling, leaving 15 kw available for distillation.

The cost of zinc production compares more than favorably with the present methods of zinc distillation, with the great advantage of treating dross on ores containing iron, copper, lead and other metals in large amount. While no attempt was made to obtain a great purity of zinc during these runs, nor using methods of refining as in my retort system, the zinc produced from the most impure zinc dross I have ever seen was more than good, as the following analysis shows:

Electric Furnace.

Charged Zinc Dross.		Refined Zinc Output.	
Lead	2.00%	Iron	.024%
Iron	6.50%	Lead	.666%
Zinc	85.50%	Cadmium	.105%
Undetermined	6.00%	Zinc by difference	99.205%

Retort Furnace.

Average Zinc Dross.		
Lead	.52%	Average zinc produced by my other process; zinc by difference 99.75 to 99.90.
Cadmium	.22%	
Iron	3.36%	
Zinc	95.90%	

From these analyses it will be seen that refining of zinc dross

into high-grade zinc in an electric furnace is a success. I believe that these runs made during July and August of 1910 produced the greatest amount of zinc metal made in an electric furnace without blue powder. As there was no change in the furnaces or condensers from their first design in 1905, there is no reason why these results could not have been obtained at that time.

Philadelphia, Pa.

The Phase Rule in Cyanidation.

BY H. B. LOWDEN.

It is not the purpose of this article to enter upon a scientific discussion of the phase rule as applied to the reactions occurring in the cyanide process, but rather to point in a general way to its bearing upon the conduct of operations in practical work. The object will be served by a consideration of the reactions between gold and an alkali cyanide if it is borne in mind that the influences at work in the treatment of silver ores by cyanidation are governed by the same laws.

Gold in ores when in contact with cyanide solution dissolves at a rate which decreases as the solution increases in precious metal content and becomes reduced in free cyanide. This decreasing rate of dissolution, analogous to the diminishing rate at which a salt dissolves in water as the point of saturation is approached, is well known, but if viewed in the light of the phase rule the reason for it is made clearer and the application of cyanide solutions to ores might in many cases be made more efficient.

Bearing in mind that dissolution will proceed until an equilibrium is reached between the undissolved metal, the metal in the solution and the free cyanide, and that as this equilibrium is approached the velocity of the reaction becomes reduced, it is easy to account for the relatively slight increase in extraction secured by prolonging treatment, as well as by the addition of fresh cyanide to solutions after being for some time in contact with the ores; the latter by reason of the fact that although the free cyanide is increased thereby, the alkali auricyanide remaining in the solution continues to exert an important influence on the velocity of dissolution. Were the alkali auricyanide removed from the field of the reaction, the velocity of dissolution would be increased by reason of the removal of that phase which, from its tendency to undergo the reverse reaction, exerts a detrimental effect by keeping the whole system near the point of equilibrium. An immeasurably small concentration of auric cyanide would suffice to bring the dissolution to a standstill were it not removed as rapidly as formed, by dissolution in the excess of alkali cyanide present with formation of the complex alkali auricyanide. Although this removal of auric cyanide is what renders the cyanide process possible, the fact should not be lost sight of that we have in the formation of alkali auricyanide only a system in which the permissible concentration is relatively large, and that while the tendency to the reverse reaction is, from a commercial standpoint, often negligible, there are nevertheless instances in which it should be given serious consideration.

A discussion of all the influences affecting these reactions, including those brought about as a result of working with such impure substances as ores, is unnecessary, and nothing would be gained by introducing complications which would only befog a clear view of the subject. The application of the phase rule points to the separation of ore from solution when the velocity of dissolution becomes low and the application of fresh solution in which the concentration of alkali auricyanide is small, and to the eventual attainment of a continuous cyanide process based on the principle of counter currents, in which the richest solution is in contact with the richest ore and the solution most free from alkali auricyanide is in contact with the ore after the dissolution of the greater portion of the values.

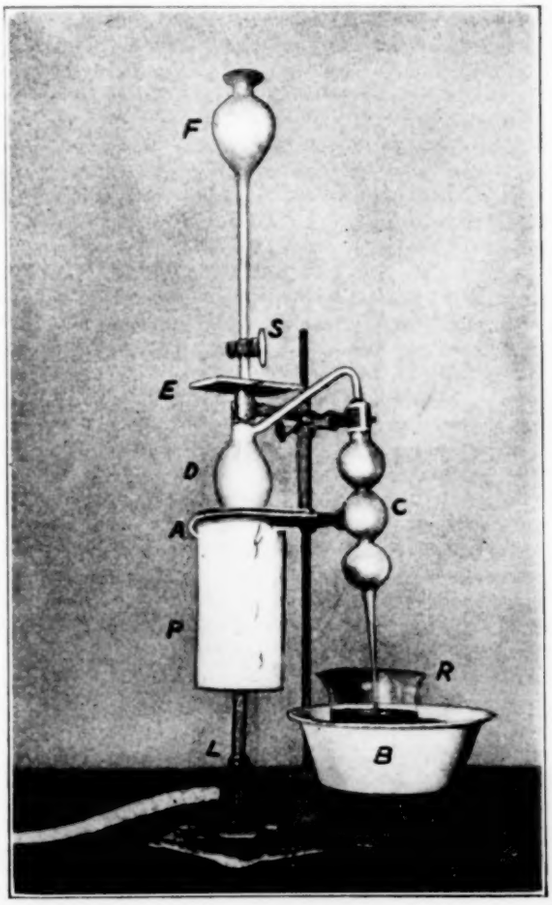
Denver, Col.

Determination of Arsenic and Antimony in Anode and Cathode Copper.

BY EDWARD F. KERN AND CHING YU WEN

In reviewing the literature on this subject, several methods were found and tried but none were sufficiently accurate for the separation and determination of arsenic and antimony in cathode copper of high purity. Some of these methods, which are in more general use, accomplish the separation of the arsenic and the antimony by distillation of the arsenic as arsenious chloride, which begins to distill at about 110° C.

These methods are in brief as follows: A sample of the copper is dissolved in concentrated nitric acid, after which about 1 gram of ferrous sulphate crystals is added, the nitrous fumes boiled off and the solution diluted. The iron, arsenic and antimony is precipitated with the addition of an excess



DISTILLATION APPARATUS.

of ammonia; thus, the arsenic and antimony is separated from the large quantity of copper, and are then separated from each other by distilling the arsenic as arsenious chloride.

These methods differ principally in the treatment of the iron precipitate, in the reducing agent used for the reduction of the arsenic, and in the procedure of the distillation.

Of these methods, the first that may be mentioned, was that which practically got rid of all the copper in the ferric hydroxide precipitate, first, by dissolving the iron precipitate with 50 c.c. of hot dilute solution of hydrochloric acid (1 acid to 5 water) through the filter, and precipitating it with concentrated NH_4OH while hot. The re-precipitated precipitate was dissolved with 20 c.c. of concentrated hydrochloric acid through the filter, and filter paper was washed with the same amount and kind of acid. This was then transferred into

the distilling flask which had previously contained 15 grams of ferrous sulphate, and distillation was made according to the method given by E. H. Miller,[1] which consisted of three intermittent distillations, using 50 c.c. concentrated hydrochloric acid each time.

The results obtained varied and were always low and showed that ferrous sulphate did not appear in this case to be a satisfactory reducing agent. Besides, the distillation, with ferrous sulphate as reducing agent, was difficult to control, because, as the solution became concentrated, it often happened that violent bumping was inevitable, sometimes so violent as to cause the distilling flask to crack.

The combined reducing agents, composed of stannous chloride and ferrous sulphate, were next employed, and the object of using the former salt was to decrease the amount of the latter. The solution containing iron, arsenic and antimony was first reduced by adding, drop by drop, a saturated solution of stannous chloride in concentrated hydrochloric acid until it became colorless, and then transferred into the distilling flask containing 7 grams of ferrous sulphate crystals. The distillation was made as before. But this also proved unsatisfactory in that the results were low, varying from 15 per cent to even 50 per cent.

The unsatisfactory results in this case were, perhaps, due to the excess and strong action of stannous chloride, which caused over-reduction of the compounds to their metallic state and these reduced metals remained undissolved even in a concentrated solution of hydrochloric acid, as black particles were observed in the flask during and after the distillation. Another objection to the use of stannous chloride was that it complicated the determination of antimony. A. E. Knorr[2] thinks that stannous chloride is not a good reducing agent to use for the determination of arsenic, because of the volatility of chloride of tin.

Both sodium thiosulphate and ferrous sulphate were also tried, singly and combined. The iron precipitate with the filter papers, in this case, was treated with 20 c.c. of concentrated nitric acid in a 350 c.c. casserole covered with a watch-glass. It was digested and evaporated on a hot plate until it became a pasty mass. After addition of 3 grams of potassium bisulphate ($KHSO_4$) and 10 c.c. of concentrated sulphuric acid, it was carefully heated over a free flame until fume of sulphuric anhydride was strongly given off. The solid mass, after being cooled, was taken up with 30 c.c. of concentrated hydrochloric acid and distilled as before, using either 1 gram of sodium thiosulphate, or a mixture of 0.7 gram of sodium thiosulphate and 7 grams of ferrous sulphate crystals.

The low result obtained by this method might be due partly to the loss of solution by spattering during evaporation to dryness and partly to the interference of the sulphur dioxide which was liberated by the decomposition of the sodium thiosulphate. A saturated solution of sodium thiosulphate, instead of solid sodium thiosulphate, was also tried and the reduced solution in the distilling flask was boiled for a few minutes before addition of the hydrochlorous acid, in order to expel any sulphur dioxide that might be present. But this procedure also failed to give satisfactory results, they being low, and varying from 20 per cent to 60 per cent.

Thus it is seen that the methods mentioned above did not give satisfactory results, so other reducing agents were tried, with the outcome that the following modified method, which proved to give accurate results, was finally adopted. It is as follows:

A sample of about 10 grams, in the case of anode copper, and from 30 grams to 10 grams, in the case of cathode copper, depending upon the amount of arsenic it contained, was weighed out and dissolved in a No. 6 beaker with concentrated nitric acid (45 c.c. for 10 grams of copper). The beaker was covered with a watch glass and warmed on a hot plate in order to hasten the dissolution of copper. When the solution was complete the beaker was removed from the hot plate and about 1 gram of ferrous sulphate added. The beaker was again heated to expel the red fume. The solution was then diluted to 400 c.c. or 500 c.c. and warmed. The iron, arsenic and antimony was precipitated with strong ammonium hydroxide. Sufficient excess was added to dissolve all the copper compounds.

The precipitate was allowed to settle, decanted, filtered while still warm and finally washed with a hot solution of ammonia (10 water to 1 ammonium hydroxide, sp. gr. 0.9). The precipitate, together with the filter paper was transferred into a No. 1 beaker covered with a watch glass, and digested gently on the hot plate with 20 c.c. of concentrated nitric acid, until practically all the red fume was driven off. (Care should be taken not to heat the solution vigorously lest some of it would be lost by spattering.)

After cooling the solution 10 c.c. of concentrated sulphuric acid was added and the beaker was re-heated on the hot plate till fume of sulphuric anhydride freely evolved. It was then allowed to cool and about 5 c.c. of a 10-per cent solution of hypophosphorous acid, or 6 c.c. of a 20-per cent solution of potassium hypophosphite was added (provided the solution did not contain more than 0.3 of a gram of arsenic).

The solution was again heated on the hot plate until all the excess of the hypophosporous acid, or potassium hypophosphite was destroyed; this was indicated by the evolution of strong fume of sulphuric anhydride. (In order to be sure of destroying the excess of the reducing agent let it fume for at least one-half an hour.)

The content in the beaker after cooling was transferred into the distilling flask (*D*, Fig. 1), the beaker was then rinsed twice with only 10 c.c. of water. The distilling flask was gently heated to boiling, with a smoky flame, in order to expel any sulphurous-acid gas that might be present in the solution. Bumping of the solution sometimes occurred and was prevented by imparting a rotary motion to the solution. It was then allowed to cool. After cooling any solution that was left in the beaker was rinsed into the distilling flask twice or three times with 30 c.c. of concentrated hydrochloric acid, making a total of not over 10 c.c. water and 40 c.c. concentrated HCl.

The apparatus, as is shown in Fig. 1, was now connected and adjusted; the receiver *R* (a No. 4 beaker), containing about 250 c.c. of water, was placed in a bath of cold water, *B*, and under the condenser, *C*, the tip of which was immersed into the water to a depth of about ⅛ in. Ten c.c. of concentrated hydrochloric acid was poured into the funnel, *F*, and allowed to run slowly into the distilling flask, *D*, through the stop-cock, *S*, in order to drive off the air in the lower part of the stem of the funnel, which would disturb the regularity of the acid-feed. The stop-cock was closed and the funnel filled with 80 c.c. of concentrated hydrochloric acid. Now heat was applied, first with a smoky flame, and when the solution began to boil was gradually increased until a proper flame was adjusted.

The hydrochloric acid in the funnel, *F*, was now allowed to run into the flask, drop by drop, at such a rate that the liquid volume in the flask might be during the entire distillation, maintained approximately constant. (The rate was, usually, three drops per two seconds.) When the acid in the funnel, *F*, was nearly exhausted. the distillation was complete. The receiver, *R*, containing the distillate was carefully removed and then the flame was turned off. The content in the distilling flask was left in place to cool and reserved for the determination of antimony.

The distillate was transferred into a No. 6 beaker and nearly

[1] "Quantitative Analysis for Mining Engineers," E. H. Miller, ed. 1904, p. 110.
[2] Private communication.

neutralized with a strong (30 per cent) solution of potassium hydroxide and the neutralization completed with a saturated solution of sodium bicarbonate, the addition of an excess of 70 to 80 c.c. being necessary. The neutralization was done by placing the beaker in a bath of cold water, in order to keep the solution cool while the neutralization was taking place. The neutralized solution was then titrated with a standardized solution of iodine (N/10.) Starch solution was used as indicator.

Precautions in the Distillation Method of Arsenic.

The analysis of arsenic by the above method requires careful manipulation. In order to do it successfully it is of much importance that the quantity of hypophosphorous acid, or potassium hypophosphite, added for the reduction should be limited to as small amount, as given in the above procedure; otherwise it reduces not only the compounds of arsenic and antimony to their metallic state, but also those of other metals, such as iron, copper, etc.

Over-reduction, according to A. E. Knorr[3], would fail to give satisfactory results, even though the metallic arsenic and antimony would afterwards re-dissolve in the concentrated hydrochloric acid solution. It is also important that complete destruction of the excess of hypophosphorous acid, or potassium hypophosphite, should be accomplished before the transference of the reduced content (solid salts and solution) into the distilling flask takes place. Any of either reducing agent remaining undestroyed will interfere with the determination and the result will in all cases be low.

The advantages of having a continuous acid-feed, during the entire distillation, are great and manifold. In the first place, the temperature may be properly regulated, secondly, the strength of the hydrochloric acid in the distilling flask may be maintained to prevent the reversible action, which may be shown by the following chemical equation:

$$As_2O_3 + 6HCl \leftrightarrow 2AsCl_3 + 3H_2O.$$

and, finally, a more rapid and effective distillation may result and, therefore, much time may be saved.

In order to secure a steady flow of the acid-feed, a grooved rectangular board of asbestos, *E*, is placed on the mouth of the distilling flask, so as to prevent the heating of the stop-cock, which would, otherwise, cause irregularity of the flow. Back-suction is the chief cause of failure. It is caused either by draught or by variation in the flame, which supplies insufficient heat. To prevent the former it becomes necessary to protect the flame from draught by suspending under the ring, *A*, an asbestos cylinder, *P*, 4½ in. high and of the same diameter of the ring. To prevent the latter, the flame should be carefully regulated from time to time. In this way back-suction is less liable to occur and success may be insured. If the distillate is once sucked into the flask, re-distillation proves absolutely fruitless, because arsenous chloride refuses to distill in such dilute solution.

The distillation apparatus was made by E. Machlett and Sons, glass-blowers, at 143 East 23rd Street, New York City. It is known as the Knorr Arsenic Distillation Apparatus.

Test of the Above Methods.

The accuracy of the above method was tested according to the following procedure: A weighed sample of about 0.1 gram of c.p. arsenous oxide was dissolved in 10 c.c. of sodium hydroxide. When the solution was complete 50 c.c. of concentrated nitric acid was added and then 20 c.c. of saturated bromine water in order to oxidize the arsenic to the higher state. The solution after an addition of about 1 gram of ferrous sulphate was heated to expel the excess of bromine, and after adding 15 grams cupric sulphate to the solution it was diluted to about 300 c.c. and heated to boiling. The iron and arsenic was precipitated with concentrated ammonium hydroxide and the red precipitate was treated in the same way as above described.

[3] Private communication.

In testing this method both potassium hypophosphite and hypophosphorous acid were employed as reducing agents. In the case in which potassium hypophosphite was used the amounts of arsenous oxide taken for analysis were 0.1004 gram and 0.1014 gram, which corresponded to 0.076 gram and 0.077 gram of arsenic respectively, and the analyses gave 0.070 gram and 0.072 gram of arsenic. In the case where hydrophosphorous acid was used the amounts of arsenous oxide were 0.1027 gram and 0.1014 gram which corresponded to 0.078 gram and 0.077 gram arsenic while the analyses gave 0.070 gram and 0.071 gram of arsenic. The values in both cases approximated the calculated values of arsenic.

Determination of Antimony.

For the determination of antimony Weller's[4] method was adopted. After the separation of arsenic by distillation the content in the distilling flask was transferred into a No. 5 beaker and diluted to about 400 c.c. The antimony, together with the copper, was precipitated in warm solution with a stream of hydrogen sulphide which continued to pass until the precipitate settled down and the solution became clear. The precipitate was separated first by decantation and then by filtration, and washed three times with hydrogen-sulphide water. It was placed with the filter paper in a No. 2 beaker and treated for an hour at room temperature with 40 c.c. of potassium sulphide (10 per cent) solution. The copper sulphide was filtered off, washed three or four times with hydrogen-sulphide water and discarded. To the filtrate now contained in a No. 5 beaker 50 c.c. of dilute sulphuric acid (1 acid to 4 water) was added to precipitate the antimony. The yellow precipitate was filtered off and washed with hydrogen-sulphide water three times.

The antimony sulphide, together with the filter paper, was placed in No. 2 beaker, treated with 40 c.c. of concentrated hydrochloric acid and oxidized to the pentad state by adding, little by little, about one gram of potassium chlorate, and the solution was heated to expel the free chlorine. (During the heating, should the solution show a dark coloration, more potassium chloride should be added.)

After oxidation and the complete solution of the antimony sulphide, the filter paper was filtered off and washed three or four times with a hot dilute solution by hydrochloric acid (1 part acid to 3 parts water). The filtrate was evaporated to 50 c.c. so as to make the solution to contain a constant quantity of hydrochloric acid. Twenty c.c. of concentrated hydrochloric acid was added and the solution diluted from 600 c.c. to 700 c.c. After the addition of 3 grams of potassium iodide crystals the solution was thoroughly stirred until the potassium iodide completely dissolved. It was then titrated at room temperature with a standardized (N/10) solution of sodium thiosulphate, starch solution being used as indicator.

As the liberation of all iodine does not take place instantaneously it is, therefore, necessary to titrate the solution slowly, that is, the thiosulphate solution should be run into the solution drop by drop until the blue color disappeared at least for one minute. Should, in any case, the blue color return immediately, more thiosulphate solution should be added.

Metallurgical Laboratory, School of Mines, Columbia University, New York City.

Metallurgy at Cobalt.—In the sixth annual report of the Nipissing Mines Company it is stated that the company has erected a small plant at the mine to test on a working scale a process devised by Mr. Charles Butters for the treatment of high-grade cobalt ores. This is in accordance with the desire of the company to find some means of reducing to bullion the silver in the high-grade ores without the expense incidental to a regular smelting plant. The present experiment is said to be satisfactory as far as tested and bullion is being shipped.

[4] "Quantitative Analysis for Mining Engineers," Miller, 1904, p. 106.

The Electric Iron Reduction Plant at Trollhättan, Sweden—I.

At the meeting of the Jernkontoret (the Swedish Association of Iron Masters) at Stockholm on May 31 a report was submitted by the engineers in charge of the electric iron smelting plant at Trollhättan, Messrs. J. A. Leffler and E. Odelberg; this report being the first official publication in regard to the construction of the plant, its operation and the results obtained during the first six months of operation.

The following abstract of this report will give an account of the construction of the plant and will in part cover the operation. In our next issue the report of the operation will be concluded and the results of the operation will be given.

There is no necessity to give here an account of the former work done in Sweden on electric iron ore reduction by the Grönwall, Lindblad and Stalhane (A. B. Elektrometall) process, as this has been fully covered in former issues and volumes of this journal. A concise historical review of the process, for which the British, United States, Mexican and other rights have been acquired by Electro-Metals, Ltd., 9½ Union Court, Old Broad Street, London, was given in our March issue (this volume, page 163). In the following we give simply from Mr. Leffler's paper an account of the experience with this process at the Jernkontoret plant at Trollhättan.

The most important points to be investigated at this plant were the folllowing:

The proper height of the furnace and other dimensions most suitable for different purposes and different charges; the most suitable design and way of building; consumption of electrodes and the shape and arrangement of electrodes; energy consumption per ton of iron of certain composition; coal consumption per ton of iron of certain composition; circulation of gas, its significance for the economy and means to accomplish circulation in case of choking charges; manufacturing of iron of different composition; conduct of various charges and concentrates and the influence of same on the economy; value of escaping gases; efficiency of the furnace; maintenance cost at the furnace.

Naturally, certain of the above questions cannot be answered definitely after six months of operation, but the experiments are to be carried on and questions which are now vague will undoubtedly have their final solution within a not far distant future. The observations made so far will greatly serve the industry as they have been made under almost ideal conditions, with greatest care and accuracy, by aid of the best instruments, and under the supervision of a highly efficient staff of engineers.

Location of the Plant.

The plant is located at Trolhättan on grounds owned by the Swedish government by agreement with the Royal Water Power Commission and will be retained until Oct. 1, 1912.

The distance from Trollhättan railway station is about 1 mile, the works being connected with this station with a normal-gage railroad track. Two sidetracks are laid to the site, one being for transport of fuel to the fuel storage house and the other for transport of other materials to the works and of iron from the works.

The fuel is charcoal, which is conveyed from the railroad trucks into the storage house by two band conveyors of cold-rolled steel. From the storage house the charcoal is elevated to the furnace top in suspended baskets running on a wire rope.

The ore is barrowed in the usual manner from ore heaps to the crusher plant, from which it is elevated to the furnace top in a truck running on an inclined track.

A narrow-gage track has been constructed for transport of iron and slag to the iron storage and slag tip respectively.

The *coal-storage house* has the following dimensions: Length, 48 m; width, 21 m, and height to the eaves, 10 m. It has a capacity of about 7600 cu. m, which approximately corresponds to six months' consumption. The charcoal is shoveled from the railroad cars into wooden hoppers under which the band conveyors run, conveying the charcoal into the storage house. The band conveyors from the hoppers to the storage house have an inclination of 1:4.4 (13°) and run in wooden troughs, the sides of which are removable so that the charcoal can be unloaded by means of a scraper at any desired point.

The steel bands are 400 mm broad and 0.95 mm thick. Each conveyor is made of three lengths of such steel bands riveted together and has a total length of about 170 m. A 16-hp motor is provided for driving, but the actual power required for one conveyor is about 7 hp and about 10 hp for both conveyors working simultaneously. The speed is 0.45 m per second, and the average carrying capacity of each conveyor is about 20 cu. m per hour.

Crushing Plant.—The dimensions of the jaws of the crusher are 550 mm x 300 mm. The machine is placed on a concrete foundation, so that the working floor is about 2.75 m above the ground level. The barrows are dumped outside the house into an elevator truck, which brings the ore up to an incline along which it slides into the crusher. The crushed material falls into a sheet-iron hopper. In this hopper a sufficient quantity is collected to fill the elevating truck to the furnace top. The door at the bottom of this hopper is operated from the working floor, and the man in charge of the crusher also regulates the elevator to the furnace top. A 40-hp electric motor is provided for driving the crusher and its elevator.

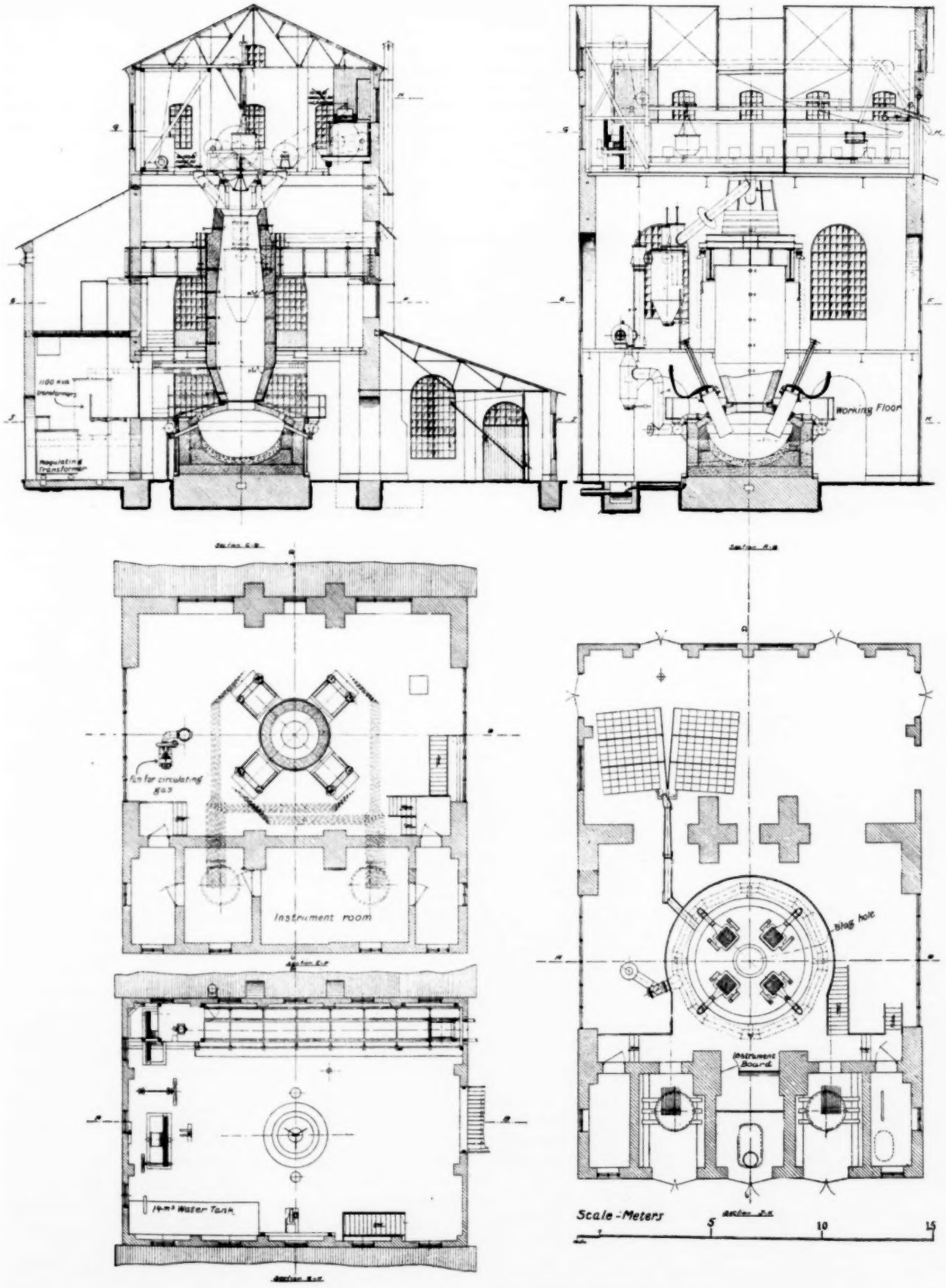
The *furnace house* has a floor space of 24 m x 15.5 m. The height from the ground floor to the furnace top is 13.7 m, and the total height to the eaves is 21 m. With regard to the electric installation, the high-tension apparatus and transformers are placed in a separate section of the building. From the two transformers for the furnace the low-tension conductors enter the furnace house proper through two openings in the wall.

The construction of the furnace is shown in Fig. 1. In this construction of furnace two separate portions are to be distinguished—that is, the shaft and the crucible or hearth. The shaft consists of a shell of steel plates 10 mm thick around the stack and 12 mm around the bosh. At the top it is riveted to an octagonal supporting ring of channel iron, on which the shaft is supported. The walls of the house on both sides of the furnace are built out at four places as suporting pillars. At a height of 9 m from the ground level they support two built-up beams, 1300 mm high, with a free length of 10.6 m. Around the furnace these are joined to form an octagon on which rests the channel-iron ring mentioned above.

The shaft is lined with Höganäs firebrick to a thickness of 360 mm in the bosh and 450 mm in the stack. At the top of the shaft is placed a Tholander charging bell and cone, which is raised and lowered by means of a capstan driven by a 2½-hp reversible three-phase motor. The cone is connected to the capstan by a 12-mm wire rope provided with a hook, which is removed between the charges so as to be out of the way of the coal baskets which pass close to the center line of the furnace.

The crucible rests on a concrete foundation. Like the shaft, it is surrounded by a sheet-iron shell 15 mm thick. At the top this shell is reinforced by a 200-mm x 18-mm iron band to take up the pressure of the arched roof. The lining of the hearth is of Höganäs and Stabbarp firebrick next to the shell and foundation with an inner lining of magnesite brick, while the dish-shaped bottom is formed of a mixture of magnesite and pitch rammed in so as to extend for some distance up along the walls. The roof is constructed of Ifö firebrick. The hearth and shaft are joined by means of a sand seal.

The four electrodes project through the roof in a slanting position at an angle of 65° to the horizontal. Each electrode is built up of four carbons of square total section of 660 mm x 660 mm. At the openings in the roof the electrodes are surrounded by cooling jackets of copper provided at the top with asbestos packing to prevent the leakage of gas. The connections to the electrodes are arranged at their upper ends and



ELECTRIC FURNACE PLANT FOR IRON ORE SMELTING AT TROLLHÄTTAN.

wedged between the electrodes and a holder of cast steel. This holder is supported on a frame movable up and down between two guides placed on each side of the electrode.

In this process gas is drawn by means of a fan from the gas outlets and blown into the crucible. Consequently the quantity of gas passing any given section of the shaft can be varied within certain limits by varying the speed of the fan.

This circulation of gas has two special objects:

a.—The gas blown into the crucible is there heated and gives off its heat to the charge in the shaft in passing up through it. Through this heating of the charge in the shaft a reduction of the ore by CO is facilitated, so that this gas is utilized to some extent for the process.

b.—The second purpose of the circulation of the gas is connected with the construction of the crucible, as the gas cools the roof and thus protects it from overheating. This cooling action of the gas is due in part to the absorption of heat, raising its temperature, and in part to the decomposition of CO_2 and H_2O in the gas in contact with the incandescent carbon in the crucible.

The arrangement of the gas pipes is shown in Fig. 1. From two gas outlets at the top of the shell the gas passes into a large dust-collecting chamber and from this chamber the circulating gas is drawn by the fan, which blows it into a circular pipe surrounding the furnace. From this pipe it finally enters the furnace through four tuyères. The volume of gas corresponding to the gas formed escapes from the dust-collecting chamber through a pipe extending above the roof of the building and burns there.

In Fig. 1 four gas pipes are shown leading into the cooling jackets around the electrodes. These pipes were, however, soon disconnected, as it was found that the electrodes were attacked too strongly by the carbonic acid in the gas blown through them. The original intention of introducing the gas around the electrodes was to cool them as well as the surrounding brickwork.

The fan was designed for a speed of 2500 r.p.m. and a consumption of 8 hp and to deliver 70 cu. m of gas per minute against a back pressure of 325 mm of water, the temperature of the gas being assumed at 200° C. and its composition to be equal volumes of CO and CO_2.

The arrangement of tap hole, feeders and pig beds is also shown in Fig. 1. The iron is conducted in trenches of a total length of about 6 m to a small swinging spout in the foundry. There are six beds arranged in two rows of three beds each containing twenty-eight pigs. Each section is provided with its distributing feeder, to which the iron is conducted through the swinging spout. For tapping of the slag the crucible is provided with a slag hole from which the slag is tapped into slag bogies of ordinary construction. During a large part of the time the charge has, however, been high in iron and the quantity of slag correspondingly small. Under these conditions separate tappings of slag have not been made, but the slag which has accompanied the iron has been skimmed off and allowed to flow into a sand trench.

Among the arrangements at the furnace top the charging arrangement and provision for lifting the cone have already been mentioned.

The ore boxes are constructed of 2-in. boards. They are eight in number, and each has a capacity of 3.8 cu. m. Their total capacity is, therefore, about 70 tons of ore. A tipping wagon runs on a track above the ore boxes and into this wagon the elevator truck discharges the ore.

Two capstans are placed on the top platform, one for elevating the ore truck from the crusher and the second for bringing up the charcoal baskets. Both are driven by one 16-hp motor.

In one corner of the top floor is a water tank of 14 cu. m capacity. Water is pumped from the Stallbacka canal close to the works. The pumping plant contains two centrifugal pumps (one as a standby), each having a capacity of 1500 liters per minute. Of this quantity 625 liters per minute are for the requirements of the plant and the remainder is at the disposal of the adjoining works of the Trollhättan Elektrothermiska Aktiebolag.

The office building is located at the south corner of the site and also comprises the watchman's room, laboratory and storerooms. On the other side of the furnace house is a smaller building which contains a small repair shop and a shop for storing and fitting the electrodes. A grinding mill is also placed there.

The electric energy for the plant is obtained from the government's power station at Trollhättan and is three-phase alternating current of 25 cycles and 10,000 volts tension.

The two furnace transformers are each of 1100 kva with a guaranteed overload capacity of up to 1375 kva under one hour. They are coupled according to the well-known Scott method, giving secondary two-phase currents. Regulation is possible between 50 volts and 90 volts in eight stages and by different connections of the secondary coils, also from 100 volts to 180 volts. The voltage can be regulated during operation.

When the plans were first made for this installation there was some doubt whether such regulation should prove satisfactory and a special three-phase regulating transformer was installed to be coupled direct before the furnace transformers and thus giving these varying voltages. As soon as this plant commenced operations it was shown that the simple regulation direct on the furnace transformers was very satisfactory and the regulating transformer has never been used. This transformer has lately been exchanged for an 1100-kva single phase transformer designed to serve as reserve for both main transformers.

The low-tension conduits from the transformers are taken through the wall into the furnace-room. These conduits consist of six copper bars, 8 mm x 200 mm, for each electrode. These bars are supported on insulated clamps of brass. The energy is conducted to the four electrodes through 192 flexible cables (forty-eight per electrode). Each cable has an area of 185 sq. mm.

The instruments required for observing and regulating the working of the furnace and electrodes, together with all handles and hand-wheels for regulating the current, are placed on an instrument board on the working floor. In the so-called furnace office on the floor above are placed the self-recording kilowatt meter (showing the power consumption for the *furnace* only) and two kw-hour meters with a maximum indicator.

In the same room are placed two galvanometers for the eight thermoelectric pyrometers which measure the temperature in the furnace at the places indicated in Fig. 1.

At the side of each pyrometer hole on the furnace is a smaller opening which was used for observing the gas pressure and for collecting samples of gas.

Ordinary U gages with colored water are permanently connected to the furnace top, the discharge side of the fan and to a hole on the roof of the crucible. A "Mono" apparatus for a maximum of 60 per cent CO_2 records the percentage of CO_2 in the gas taken from the discharge side of the fan and the pressure is registered by a recording gage. The pressure due to the speed of the gas returning to the furnace is measured at an opening on the suction side of the fan, where the temperature of the gas is also determined by a mercury thermometer. The "Mono" apparatus, the gas calorimeter, the recording-gas pressure gage, the speed gage and a gas-analysis apparatus of Sondén's type are placed in the room close to the furnace office, and are there easily accessible for the engineer in charge, who usually remains on the operating floor.

In electric pyrometers platinum-platinum-rhodium elements and silver-constant elements are used, the former in the lower and the latter in the upper part of the furnace. They are arranged to record the temperature every ten minutes. For

determining the temperature of the iron and slag on flowing from the hearth an optical pyrometer of Siemens & Halske's make is used. For the direct determination of the calorific value of the gas a gas calorimeter of Dr. H. Strache's, Vienna, construction is used. The quantity of gas passing through the gas pipes is determined by means of Krell's pneumometer and micromanometer. It has so far been impossible to use the recording instrument for determining the speed of the gas owing to the soot and dust carried over by it.

Cost of Construction.—In the estimate of Oct. 16, 1909, the total cost of the works was estimated at $60,700, and the association appropriated a sum of $69,000 for the installation.

The accounts show, however, that the actual cost amounted to $88,500. The excess in cost was caused by the whole construction being carried out in a more elaborate and substantial manner than was at first contemplated. The furnace house, for instance, was first intended to be constructed partly of wood, but has been built entirely of brick and steel construction, and the furnace shaft instead of being supported on a steel framework from the ground is supported on heavy beams resting on the brick walls.

Management of the Works.—The management of the plant is in the hands of a committee appointed by the association and consists of Major-General G. Geijer, Mr. Aug. Herlenius, chamberlain to H. M. the King, and Messrs. J. A. Brinell, chief engineer of "Jern-Kontoret," Axel Wahlberg and E. Odelberg, the last named acting as managing director for the committee.

At the works the management has been in the hands of Messrs. J. A. Leffler and E. Nyström, both engineers on the staff of "Jern-Kontoret," the former acting as general manager and the latter as superintendent.

The employees at the furnace consist of three men for charging, two for bringing up charcoal, four for crushing and bringing up ore, five furnacemen, three men for fitting electrodes, two for weighing iron and slag, one blacksmith, one blacksmith's helper, one electrician, five men for loading and unloading, one carpenter and one watchman and messenger.

The report will be concluded in our next issue, when the following subjects will be taken up: The principles of the reduction furnace; principles employed for calculating the form and dimensions of the shaft; charcoal, ore and limestone; electrodes and electrode consumption; the charges; the durability of the crucible and roof; iron and slag; composition and pressure of the furnace gas, and notes relating to working results.

Electric Furnaces for Molten Materials.

BY CARL HERING.

To design an apparatus as perfectly as possible one should first find what the desired ideals are and then endeavor to reach them as closely as practicable and in proportion to their importance.

The chief desirable ideal features of a commercial electric furnace for molten charges seem to be the following: Greatest possible economy in power; greatest possible rapidity of treatment, hence greatest tonnage per day for a given furnace; heating from the bottom instead of from the top; no higher temperature than that actually required; rapid and uniform distribution of the heat; rapid circulation to produce greatest possible homogeneity of product; largest possible surface of contact between the metal and the slag when the treatment is dependent upon the chemical action of the slag; heat transmitted from the metal to the slag rather than the reverse; greatest possible reduction of the troubles, cost and power losses due to the electrodes possibility of adding fuel heat; ease of control; reliability of operation; simplicity and cheapness of construction and operation.

The economy of power involves numerous features. The ideal method of heating is to generate the heat in the material itself, where it is wanted, as distinguished from generating it elsewhere and transmitting it to the material by radiation or conduction. There will then be no waste, as the efficiency of the utilization of the heat will be 100 per cent—that is, theoretically perfect. It is done in this way in the induction furnace, as also in all other resistance furnaces in which the material treated is itself the resistor. In the arc furnace whatever heat is generated in the arc itself and in the electrode can reach the material only by radiation and then only when the temperature is higher than that of the material.

To always pass the current through the whole charge means either a constant depth of bath, which excludes feeding with solid material, or else it means a provision for the regulation of the current through very wide limits, hence large and expensive machinery. For these reasons it seems preferable to pass the current through only a small portion of the charge, which portion is always constant and is independent of the depth of the bath. But this becomes practicable only when this small heated portion is continually and rapidly removed as soon as it is heated and is replaced by cooler material, otherwise it would soon be volatilized, which would break the circuit, injure most linings and possibly produce explosions. In other words, this is operative only when there is a circulation through the channels containing the portion of the charge which is being heated, and if such circulation is not very rapid, the furnace will necessarily either be a large slow-acting one, or the heated material will get too hot for the lining; circulation by gravity due to the heated material rising would be far too slow for a commercial furnace. The lack of means for producing a rapid circulation through such restricted resistor channels seems to have been the reason why this very desirable method had not come into more general use.

To further increase the heat economy the top surface of the charge and that exposed to the walls should be as small as possible for a given charge, as the losses of heat from an internally heated charge are due entirely to these surfaces and to the electrodes; they are less the smaller these surfaces are. The ideal shape of the charge would therefore be a sphere; this, of course, is not practicable, but the shape should approximate to it as closely as possible, say, by making the charge the shape of a hemisphere if practicable instead of having the hearth large, flat and shallow.

These losses from the surfaces of the charge seem to be far greater than is generally supposed, hence in addition to reducing these surfaces as much as possible the wall thicknesses should be made as great as practicable, and of the best heat insulating material; bricks are cheaper than kilowatts, and they represent only a first cost, while the kilowatts lost are a continuous cost. There should also be as many joints between bricks as possible, transversely to the direction of the flow of heat, as joints seem to be excellent and cheap heat insulators and take up practically no space. The usual heat insulation of combustion furnace is very crude and wasteful and would be very inadequate for electrical furnaces in which those constant losses have a far greater money value.

Still further heat economies will be obtained the more rapid the treatment; these losses through the walls and electrodes evidently continue during the whole time of treatment, hence if its duration is reduced these important losses will be reduced in the same proportion. Hence it is very important that the heating should be done as quickly as possible and that everything possible should be done to hasten the various actions which bring about the melting and refining. If, for instance, it takes 50 kw merely to keep a ton of steel at the desired temperature during refining, which means that these losses amount to 50 kw per ton, then if the time necessary for treatment could be hastened so as to be only half as long, for instance, the energy required (in kw-*hours*, not in kilowatts) would evidently be only half as great. It is in this increase in rapidity of treatment that probably the greatest improvements in the reduction in power will be made in the future. It increases the

tonnage *per day* delivered by a given furnace. A small, rapidly acting furnace could produce the same tonnage per day as a large slow acting one, and the leakage losses of heat will be less per ton.

Attention need hardly be called to the fact that, as most hot materials rise it is far better, more rational and much more rapid to heat from the bottom instead of from the top whenever practicable. Heating a liquid by means of the arc, or by means of directly impinging gases, as in the open-hearth furnaces, must necessarily always be done from the top, hence the rapidity of the heating is then limited by the necessarily slow conduction of heat from the top to the bottom. Cases have occurred in which pieces of metal in the bottom of the charge were still unmelted, although the refining of the top of the charge had been completed. The ideal method is unquestionably to heat from the bottom. This becomes possible with some types of resistance furnaces.

Owing to these continuous losses, which are by no means small, the temperature ought not to be any higher than that required for the desired reactions, as these losses increase when the temperature rises. In the resistance furnace the temperature is generally within one's control, while with the arc it is the boiling point of the electrode material, hence generally far higher than required and the radiation is also to the sides and roof, where it is not only not wanted and is lost, but where it sometimes even acts destructively.

The more rapidly and uniformly the heat is distributed throughout the mass the more rapid will be the treatment and the less will be the leakage losses, as the temperature will be lower and the time of treatment less.

Homogeneity of the product is often of prime importance, particularly in steel refining. Complaints have been made that the metal at the bottom of a hearth of an arc furnace sometimes has a very different composition than that at the top. Moreover, an overtreatment of the metal at the top in order that after mixture with that at the bottom it will have the right composition, is a makeshift, and involves the element of chance to a great degree. Hence, circulation is very desirable.

Moreover, materials are already in use which when introduced into molten steel cause some particular refining action, and it is quite likely that more such materials will be forthcoming. When quickly and thoroughly mixed throughout the whole mass the time necessary for the reactions should be proportionately reduced, thereby decreasing the losses of heat, increasing the tonnage of a furnace per day (or reducing the size of a furnace for a required daily tonnage) and delivering a more uniform product.

The same applies to the alloying materials which are sometimes introduced into the molten metal; some of these tend to float and others tend to sink, hence circulation is very important.

When the refining is dependent upon the chemical action of the slag, as in steel refining, it is evident that such an action can take place only at the surface of the contact of the metal and the slag. Hence, the time necessary for such refining would be less the greater this surface. To have such large surfaces in a bath in which the renewing of this surface is accomplished chiefly or only by gravity would mean that the continuous losses of heat would be relatively very large. It would, therefore, be far more desirable to have the dimensions of this chemically active surface small and then increase the actual and effective surface by continually renewing it as, for instance, by circulating the metal rapidly in a more or less vertical direction up against the blanket of slag.

Rapid circulation is, therefore, of the very greatest importance in such electric furnaces for numerous different reasons.

It is claimed by steel makers that steel must be left standing in a quiescent state for quite a time while in a molten condition; this is called "killing" the steel. It is, therefore, alleged that circulation would be bad. The writer believes, however, that what is accomplished by this killing is to give the lighter impurities, such as gases, slag, oxides, etc., an opportunity to rise to the surface by gravity. If this is correct, the same ends ought to be accomplished far more rapidly and, therefore, more cheaply, by a forced systematic circulation from the bottom to the top (as distinguished from a mere agitation) during which all the impurities and gases would in turn be brought to the top into contact with the slag and when they have once gotten there it is presumable that they will stay there. Hence it is believed that, even though circulation is the direct opposite to resting, it will probably be found to accomplish the same object and do it far more quickly, thoroughly and cheaply.

Slag reactions require at least a certain temperature of the contact surface. This temperature is more easily maintained if the heat travels from the good conductor, the metal, to the poorer conductor, the slag, than the reverse, and the maintenance of this temperature is no doubt a very important factor in reducing the time of treatment, which in turn reduces the losses of heat per ton and increases the tonnage per day. Hence the metal should be hotter than the slag, which again becomes possible in the resistance furnace. This also reduces the maximum temperature required to maintain this necessary temperature at the surface of contact of metal and slag. If the slag is hotter than the metal, as in arc, open-hearth, or other furnaces heated from the top, the cooler good conducting metal will more quickly abstract heat from the layer of slag immediately in contact with it, forming a cooler layer of slag just where the heat is most needed; and as slag is a poorer heat conductor, it will take longer time for heat to pass down to reheat this cooler layer; hence, the action would be slower, or else the temperature of the slag must be made very much higher so as to hasten this downward flow of heat through a poor conductor. Moreover, most hot liquids are less dense than when cooler, and therefore heated liquids rise naturally by gravity. To cause heat to flow downward from a hotter to a cooler layer one must rely entirely on the very slow process of conduction, as gravity will not aid such transmission the way it does in the other direction. Rapid circulation, of course, will hasten such a transmission of heat.

The ideal way of overcoming the troubles, cost and loss of power due to the electrodes is, of course, to dispense with them entirely, as in the older forms of induction furnaces. But this also has its disadvantages, and it is significant that in what appears to be the most successful form of induction furnaces electrodes have been added.

If, therefore, these electrodes cannot be dispensed with, the next best thing to do is to make them as unobjectionable as possible. If made of the metal to be treated, their size and the losses in them become considerably less than if made of carbon or graphite, as the writer showed by actual tests some years ago. This becomes possible with resistance furnaces, but not with those using the arc. Moreover, when thus made of metal, if they are immersed under the liquid metal they cannot be consumed, or break off; they then require no adjusting and do not contaminate the molten charge. Hence in such furnaces the cost of electrodes consumed, which in arc furnaces is quite large per ton, would be entirely eliminated.

The possibility of adding fuel heat is often very desirable, especially when cold materials are charged, as the lower temperature heat is generally more economically obtained from fuel. Such heat would also reduce the loss of heat from the top surface of the electrically heated metal, would assist in melting the slag forming materials, and would lessen the amount of heat which would otherwise have to flow from the metal into the slag. The ideal form of furnace would, therefore, seem to be one in which gaseous fuel heat is supplied over the top and electric heat in the charge itself.

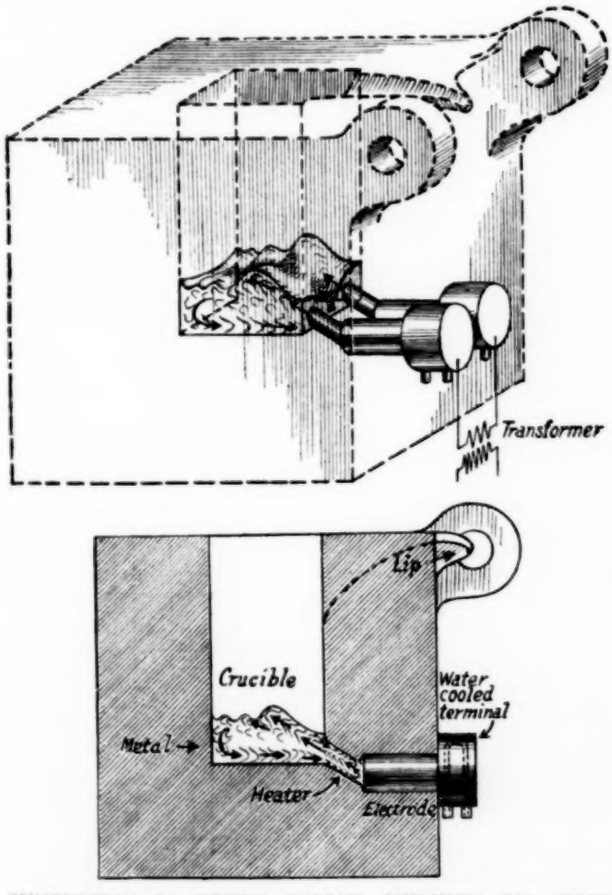
The control of resistance furnaces may generally be accomplished by the simple process of regulating the voltage of the transformer, and this can be done, within reasonable limits,

without appreciable losses of energy; hence it also approaches the ideal.

The reliability of operation, and the cheapness and simplicity of construction and operation, depend, of course, on the particular design, and, of course, the designer will approach the ideals in these features as closely as he can. In general, however, it may be said that resistance furnaces are simpler in their operation than arc furnaces, and that cheapness of first cost and of operation are good properties only when not purchased at too high a sacrifice of other desirable features.

When electric furnaces are designed so as to approach these ideal conditions more closely it is not unreasonable to predict that relatively large furnaces may become available even when the electric current must be purchased from the city mains, when all the accompanying advantages are duly considered.

With the above mentioned desirable features in mind the writer has endeavored to devise a form of furnace which



CONSTRUCTION OF ELECTRIC FURNACE EMPLOYING THE PINCH PHENOMENON FOR AGITATION OF THE CHARGE.

would embody as many of them as possible. To do this some radical departures had to be made from existing forms and types. The underlying principles of a type of furnace whose properties seem to approach some of these ideal features, especially that of rapid circulation, were described by the writer in a recent paper.[1]

Furnaces of this type are now being developed and will soon be undergoing commercial tests. The general form is shown in outline in the accompanying sketches, which represent, in its elements only, a small crucible tilting furnace.

The underlying principle, in which it differs from the usual types, is a new electromagnetic phenomenon which might be called a valveless pump. It follows as a consequence from the now well-known "pinch phenomenon" described by the writer some years ago, that when an electric current of suitable amount is passed lengthwise through a column of a liquid conductor confined in a submerged hole or tube the liquid will be ejected from the middle of this tube quite forcibly by this peculiar electromagnetic action, and at the same time will be sucked in around its circumference.

If, therefore, these holes or tubes are so proportioned that the necessary heat for the furnace will be generated in the metal in them due to its resistance, and that the ejecting force will be great enough to eject it as fast as it is heated, it will be seen that the metal in the crucible above the tubes will not only be duly and rapidly heated, but will at the same time be projected upward against the blanket of slag and be thoroughly and very actively circulated as it passes repeatedly into and out of the heating tube or resistor.

The sketches explain themselves. The heaters or resistors are the short, small tubes in which this electromagnetic force acts and in which all the heat is generated. The arrows show the circulation approximately. The electrodes, preferably made of the same metal, are the larger embedded cylinders, and are provided at their outer ends with water-cooled terminals, to which the leads from the transformer are attached. The direction of this squirting action is independent of the direction of the current, hence either direct or alternating currents may be used. The actions in this furnace have been described in the paper above referred to and need, therefore, not be repeated here.

It may, however, be of interest to add here that the ejecting force, even in a small furnace to which this principle is less well adapted than to larger ones, is great enough to cause considerable agitation, about as shown in the sketches. The suction in one case was even great enough to suck in air with a whistling noise, through a leaky tap hole at the bottom of one of the resistors against a hydraulic head of about 6 in. to 8 in. of molten lead. In another case, when the level of the liquid in the crucible had gotten too low, being nearly down to the top edge of the tubes, pieces of floating charcoal were sucked into the tubes accompanied by a snorting sound due to air being sucked in with it. When the depth of metal is sufficiently great, as it should be when operating normally, the action is entirely noiseless.

Each particle of metal is in the tube apparently for only about a second or two, judging from the velocity of the circulation. By using two tubes for each electrode the metal may be caused to pass into one and out of the other, instead of flowing in both directions in one tube.

The transformer may be directly attached to the furnace, so that the furnace as a whole may be arranged to tilt. The one shown tilts around the nose as a center, so as to enable the metal to be poured directly into the molds, thereby avoiding the use of pots.

[1] A New Type of Electric Furnace. Read at the meeting of the Amer. Electrochem. Soc., in April; abstracted briefly in this Journal for May, p. 277.

A Continuous Cyanide Treatment Method for Gold and Silver Ores.

BY JOHN E. ROTHWELL.

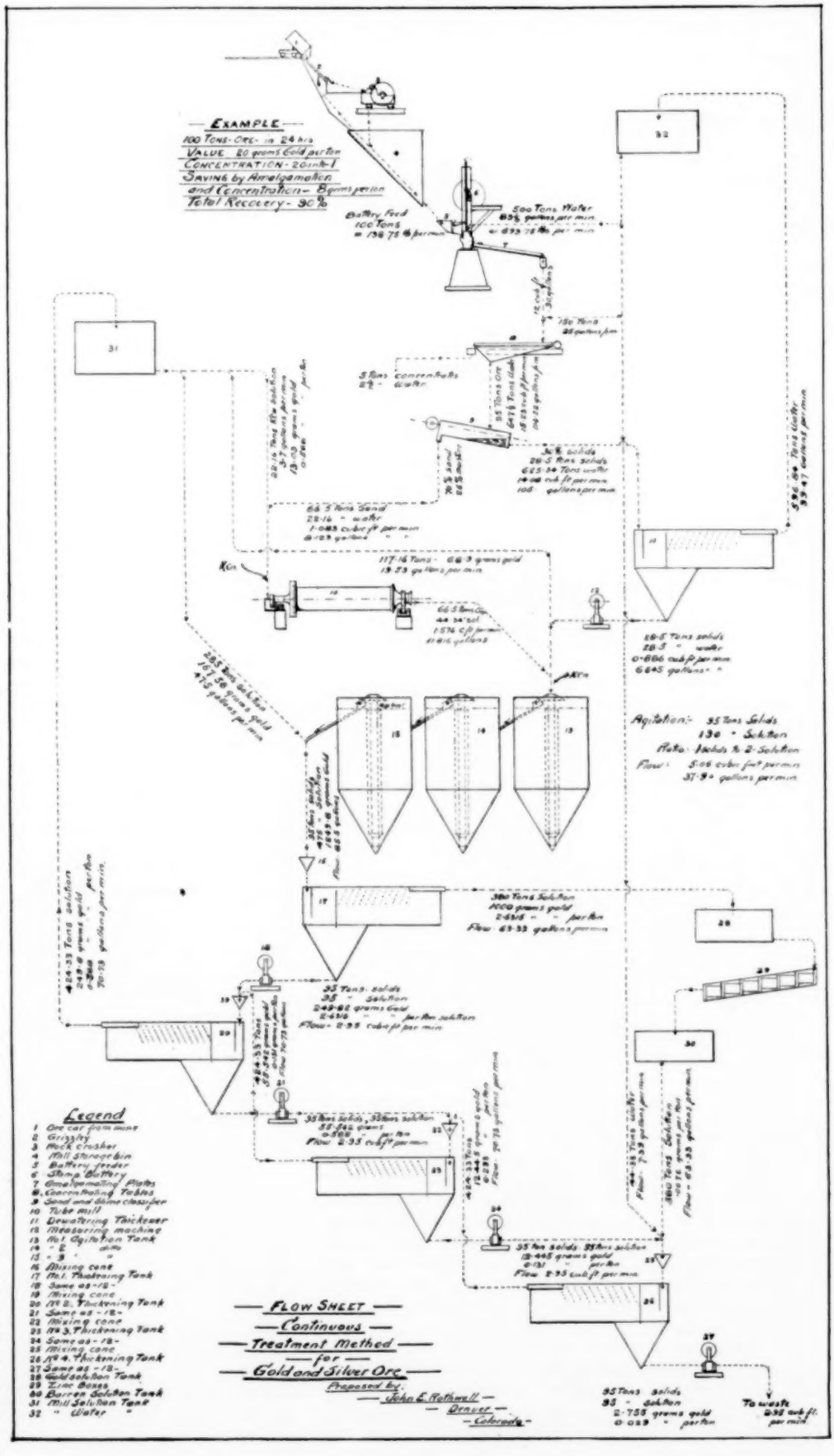
The method of continuous treatment outlined in the following is suggested as a means of operation along lines that will tend to eliminate as much as possible the personal equation which always plays such an important part in all milling operations, and bring the whole treatment to a nearly automatic operation.

Reference is made to the flow sheet that accompanies this paper, on which is shown the general machinery and equipment of a milling plant and the flow of the ore through same. Certain arbitrary conditions are assumed in the illustration, but these conditions are well within the limits of ordinary practice.

The example of milling treatment is taken at 100 tons per day, so the conditions prevailing at different points during the flow can be carefully noted in cubic feet and gallons per minute flow during the daily operation. Thus we get an idea of the volume passing certain points in a given time and the requirements in equipment necessary to handle these amounts. The mechanical automatic regulation and control are matters on which I will not go into the detail, but will say that equipment designed to perform the necessary functions, and register, by record, the actual conditions prevailing at the different points throughout the operation, is entirely a matter for the mechanical and electrical engineer, and presents no serious obstacle in either field.

The mechanical operations outlined in the flow sheet for the preliminary mechanical treatment of the ore down to the continuous agitation tanks, shown as 13-14-15 in flow sheet, are based on common practice. Some of the features shown may not be required on certain classes of ore, the leaving out of any one of which will not affect the general flow.

At the agitation tanks, it will be noted that the continuous flow is carried on through a sampling vane located *above* the pulp level in the tank. The object of this is that these tanks also may act as an equalizer for the flow of the ore through the plant. Thus, for illustration, in event of a stoppage of the milling end of the plant for a short period of time, the tank 13 can be drawn on for about one-third of its capacity and maintain a constant flow through the others; similarly the other storage tanks can be drawn



on for a part of their contents and deliver a constant flow to the cycle of operations following.

The flow of the solids after leaving the agitation tanks and the recovery of the dissolved value is based on the idea of a countercurrent flow of diluent against the downward flow of the solids and accompanying dissolved values.

The critical point in this whole operation requires the absolute control of the ratio of the two currents in order that dilutions may be positive. The slight variations of value in gold and silver to be recovered due to the change in the grade of the ore being treated will not require any radical change in the flow in either direction.

The value of the countercurrent lies in the constant change of solvent as well as diluent, the tendency of which is to avoid reaching the chemical equilibrium between the solvent and the metal to be dissolved and obtain for the solvent its highest efficiency.

The recovery of the dissolving chemical (the cyanide) when, due to economic features, a solution is used that requires this recovery, is not shown in the flow sheet; but if we follow out the general features of the recovery of the gold values, in a subsidiary treatment, this recovery can be made to any degree required for economical practice. Absolute accuracy is not claimed for the figures shown in the flow sheet, but they are well within the lines of general practice and are given to show the possibilities of the method of treatment carried out in a continuous manner.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute

The spring meeting of the Iron and Steel Institute was held at the Institution of Civil Engineers, London, on May 11 and 12.

The discussion on the paper by MM. F. A. Daubinè and E. V. Roy entitled "A Process for the Desiccation of Air by Calcium Chloride" (which is printed in full elsewhere in this issue) was opened by Mr. J. H. Harrison, who said that the value of the paper lay in its being a practical contribution dealing with iron and steel manufacture. Some six or seven years ago while the principle propounded by Mr. Gayley's paper on desiccation of furnace-blast air met with some degree of acceptance, his conclusions were received with incredulity if not with ridicule, but now it was recognized that dessication effected a saving of from 10 per cent to 20 per cent in coke. The author's method seemed to be a distinct improvement on the processes previously employed.

Mr. H. Hawdon remarked that in England very few works were equipped with apparatus for washing the waste gases used for drying the calcium chloride, and the heat required would involve considerable cost; and as to cooling apparatus, he thought that a large proportion of the chloride would not come into contact with the water at all.

Mr. J. E. Stead said the desiccation of blast by calcium chloride was first tried fully twenty-five years ago, but there was not then any provision for the obviously necessary cooling of the chloride after it had been dried. The hot blast was sent through the chloride to dry it, and was immediately followed by the passage of the cold blast. That process was, of course, ineffectual, but the author seemed to have met the difficulty.

Mr. F. W. Harbord doubted if the process would in practice do what the authors claimed in keeping surface temperature low. It was probable that as the surface of the chloride liquefied the resulting mass would greatly impede the passage of air through the grids, but if the process were really adaptable to the conditions of every-day work it would be better than the freezing process.

Mr. Lawrence Sterne expressed a desire for definite figures of working costs. The results obtained by refrigeration were known, and simplification of the apparatus with reduction of prime cost and upkeep were progressing favorably.

The balance of the report of the Iron and Steel Institute meeting will be given in our next issue.

"The Times" on Steel Prices

A special article on American Steel Prices and the effect of cutting on British Trade in *The Times* of May 31st is apparently written by an English optimist. The writer discusses the effect of the breakdown of the price maintenance understanding in the United States Iron & Steel trade, and commences by pointing out that "In the United States there are plenty of blast furnaces and rolling mills to undertake about nine-tenths of the world's export business in iron and steel, in addition to meeting home demands. If our United States rivals are able and willing to cut their export prices low enough—just a little below British and German figures—they can easily secure the supreme position in neutral markets, almost wipe out all competitors, and, besides practically destroying nearly all their rivals' export trade, they can make very serious inroads upon the purely domestic business of British steel manufacturers.

"There is a world's export trade in iron and steel of something like 13,000,000 tons a year. Of this total Britain and Germany between them are dividing nearly 10,000,000 tons; America is taking only one and a half million tons. But America has plant enough to do an export trade larger than that of Britain and Germany put together. In minerals, mills and men the United States producers are equal to the task. The only difficulty is that of price. Can our American rivals sell cheaply enough to beat us in the race?"

After offering explanations of the non-fulfilment of the repeated intimations during the last ten or fifteen years of a great export campaign by American steel manufacturers, he proceeds:

"Given the collapse of the American understanding and the prospect of an all-round reduction to the lowest profitable level, involving something like free international competition, how would matters shape themselves? On this point it is impossible to appeal with much confidence to past history. It is known that in the past American competition has been more imaginary than real. Our United States rivals have never approached us in the exportation race. . . . Is America at last about to make her long-threatened bid for the lion's share of the world's export trade in steel?

"This brings us to the root question. Can America produce at a sufficiently low cost to conduct a big export business for any great length of time? Are the costs of production really lower in the United States than they are in Britain or Germany? Twelve years ago it was authoritatively estimated that the cost of producing steel-making pig iron was less in America than in this country by very nearly 20s. per ton, and that in the case of steel rails the difference was 29s. . . . But it is a remarkable fact that since that time productive costs have gone up more substantially in the American iron and steel group of trades than in any other important branch of industry anywhere. The latest reliable figures are those supplied to the Ways and Means Committee two-and-a-half years ago, and they show that between 1899 and 1908 the net cost of producing pig iron increased by 24s. per ton, and of steel rails, 27s. per ton. This in spite of the mechanical and metallurgical improvements effected. Since the late nineties mining royalties have risen; mining costs have increased, while the yield of the ore has declined; cost of coal and coke has gone up; transit charges on water and rail have increased, so that it is not now possible to produce iron and steel with anything like the economy of a dozen years ago. There has been no corresponding rise in our costs of production for the rise in prices of some raw materials has been just about offset by improvements in processes."

One has got so used in this country to depreciation of our industrial potentialities that this article is quite a refreshing change to the much advised British manufacturer. But whether it will be safe for the lion to emulate the ostrich and hide his head in the sand in another matter. Wolf has been cried so often that the sheer novelty of a whisper of safety may have the same effect as the first of the alarms which disturbed our equanimity and make us more wary than we have recently been.

Market Prices.

May, 1911.

Copper was lower during the first part of the month, reaching £53.15.0 by the 10th, and remaining there till the 15th. It then rose steadily to the 23d, when £55 was reached. Closes at £55.

Tin was nervous at the beginning of the month, going between £195.10.0 to £192.10.0 before rising, which commenced on the 10th whence it soared pretty steadily up to its present price of £203.

Lead is slightly stronger at £13.7.6, rising on the 8th.

Haematite is down, having fallen away from 8th to 12th, and now standing at 61/9.

Scotch Pig was rather flat during the opening fortnight, touching 51/10. It commenced to rise smartly on the 18th and finishes at 52/3.

Cleveland has been much in sympathy with Scotch pig-iron and dropped to 45/11½ by the 10th. It has since recovered to 46/2.

Rubber has been low and weak, dropping from 5/- on the 10th to 4/4½ on the 17th. It has since become steadier and closes at 4/2.

Alum, lump, loose, per ton	£5.12.6
Antimony, black sulphide powder, per ton	22. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper ore, 10 to 25 per cent., per unit	9/1½ to 9.7½
Copper sulphate, per ton	22. 0.0
Carbolic acid, liquid 97/99 per cent, per gal	1.4
Creosote, ordinary good liquid, per gal	0. 0.2
Camphor, 1 oz. tablets	1.9
Caustic soda, ash 48 per cent, ordinary, per ton	5.10.0
Litharge, flake, per ton	15.10.0
Mica, small slab, original cases, per pound	6d. to 2.0
Naphtha solvent, 90 per cent. at 160° C., per gal	0. 0.9
Petroleum, Russian spot	0. 0.5⅛
Sal ammoniac, lump, firsts, delivered U. K., per ton	42. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	13. 5.0
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N., orange spots, per cwt.	4. 4.0
Tin ore, 70 per cent., per ton	£125 to 127. 0.0
Hydrochloric acid, per cwt.	5.0
Zinc, sheets, Vieille Montagne, f.o.b. Antwerp	29.10.0
Quicksilver, per bottle	8. 5.0
Platinum, per oz.	8. 6.6

On the month the following differences are noted:

Higher.	Amount.
Copper	£1.0.0
Tin	9. 5.0
Lead	3.9
Copper ore, per unit	0.1½
Copper sulphate	2.5.0
Tin ore	2.0.0
Carbolic acid	0.0.1

Lower.	
Haematite	0.1.6
India rubber, Para	0.1.1
Scotch pig	0.0.6
Cleveland warrants	0.0.4

London, England.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Electrically Refining Molten Pig Iron.—In our March issue, page 127, we published an article by Mr. J. B. Nau on the preliminary refining of pig iron, more especially of phosphoretic pig iron too high in silicon to be treated directly and economically in the open-hearth furnace. The essential principle is to submit pig iron, running from the blast furnace, to the refining action of solid pieces of iron ore. Silicon, phosphorus and carbon will thereby be eliminated, but in order to maintain the required temperature it will in general be necessary to supply heat to the charge during the process, as was shown by the heat-balance sheet in the article referred to. For this reason Mr. Nau proposes to carry out the process in an electric furnace, as it is the easiest way of controlling the temperature absolutely. Three recent patents of Mr. Nau refer to this process. From the last two we reproduce herewith the

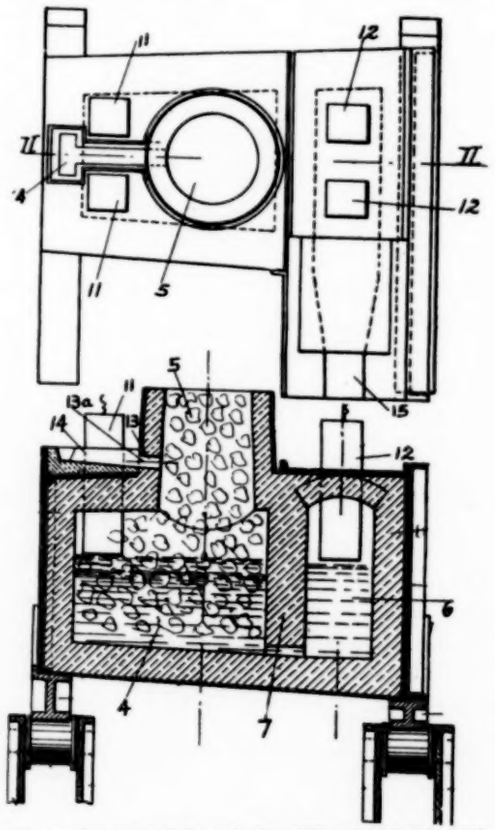


FIG. 1.—ELECTRIC FURNACE FOR REFINING OF PIG IRON.

tilting furnace in which the refining is carried out. As shown in plan and vertical cross-section in Fig. 1, it consists of two parts, a crucible 4 surmounted by an ore shaft 5, and a forehearth 6 separated from the crucible 6 by partition 7 and communicating with it through holes 8 near the bottom. The forehearth 6 is provided with an outlet spout (not shown in the illustrations) for pouring the refined metal when the furnace is tilted. 11 and 12 are the electrodes. After having heated the empty furnace by gas, preheated rich iron ore in solid pieces or in the shape of concentrate ore briquets is introduced into the crucible 4 through the ore shaft 5. The liquid pig iron from the blast furnace, which is to be refined, is poured into spout 14, enters the ore shaft through opening 13, percolates through the ore and passes over to the forehearth 6. Otherwise the illustration is self-explanatory. (984,590, Feb. 21; 990,765, April 25; 991,633, May 9.)

Electric Iron Ore Reduction.—An electric furnace specially adapted to the electric smelting of iron ores by a con-

tinuous process is described in a patent of Mr. EDWARD R. TAYLOR under Electric Furnaces.

Nickel Steel with the Aid of Electrolysis.—See the abstract of a patent of Betts for electrolytically refining alloys under Electrolytic Processes.

Electric Furnaces.

Electrode Holder.—A new construction of electrode holder of electric furnaces, due to Dr. PAUL L. T. HÉROULT, of La Praz, France, is shown in Fig. 2. Diagram *I* shows the carbon electrode *A* attached to the carbon holder *B*, which is constructed as a clamp encircling the carbon. This ring may be

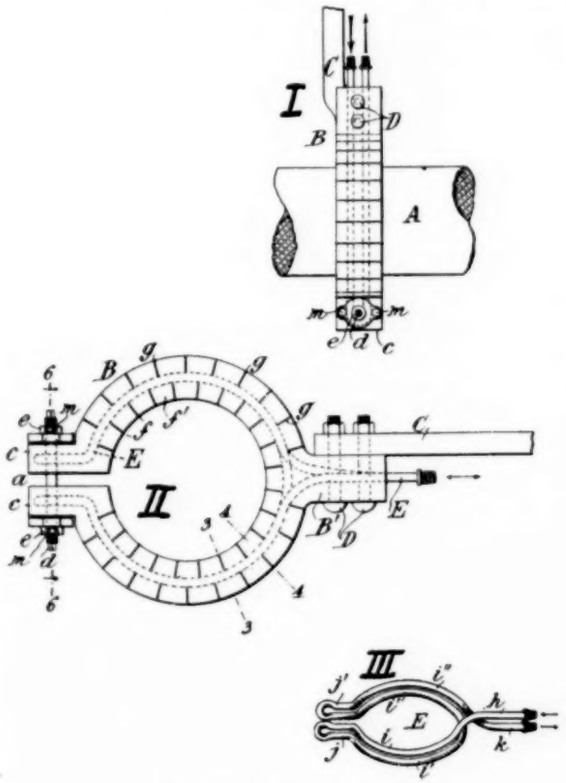


FIG. 2.—ELECTRODE HOLDER.

divided on one side *a*, as in diagram II, or at oposite sides. At the division *a* the clamp is formed with flanges *cc* through which passes a tightening screw *d*. To render the holder sufficiently flexible to readily adapt itself to slight irregularities in the surface of the carbon, it is slitted radially on its inner side at *ff* and on its outer side at *gg*, the outer and inner slits being staggered or alternated. The electric current connection is through the bar *C*, which is clamped to the arm *B'* by bolts *D*. For water-cooling a duct *E* is provided, consisting of an iron tube of the shape shown in diagram III. The main portion of the holder *B* is of copper or brass and is cast around the iron tube *E*, so that the latter is entirely embedded. (989,148, April 11, 1911.)

Electric Resistor Furnace.—The resistor furnace of Thomson and FitzGerald, the characteristic feature of which is heating of the charge by radiation from resistors on top of the furnace, and its application to zinc smelting have been repeatedly mentioned in this journal. To reduce the heat loss due to radiation Mr. JOHN THOMSON has now modified the construction by embedding the carbon resistor (which is otherwise of the same construction as before) in the charge itself. In order to prevent the electric current from leaking into the charge and in order to protect the resistor at the same time it is surrounded by a casing or shield. This casing is not in direct contact with the resistor, but there is an air space between them, which may be filled with an inert gas instead of air. For the construction of the casing fireclay, "crucible mixtures" (combinations of graphite and clay) and recrystallized silicon carbide may be used. (994,217, June 6, 1911.)

Titanium as Reducing Agent.—Mr. JOHN B. HUFFARD, of the Electro Metallurgical Company, of Glen Ferris, W. Va., produces alloys of titanium with vanadium or molybdenum, etc., by reacting on an oxide of the alloying metal (vanadium, molybdenum, etc.) with a titanium reducing agent, for instance, metallic titanium or titanium nitride, carbide or monoxide, etc. The titanium reducing agent is produced by the reduction of rutile; it may contain metallic titanium and relatively small amounts of nitrogen, carbon and oxygen. (992,423, May 16, 1911.)

This same titanium reducing agent is employed by the same inventor for the production of the metals tungsten, vanadium, molybdenum, etc., free from carbon. With tungstic acid the reactions are as follows:

$$2\,WO_3 + 3\,Ti = 2\,W + 3\,TiO_2.$$
$$2\,WO_3 + 3\,TiN = 2\,W + 3\,TiO_2 + 3\,N.$$
$$WO_3 + 3\,TiO = W + 3\,TiO_2.$$
$$WO_3 + TiC = W + TiO_2 + CO.$$

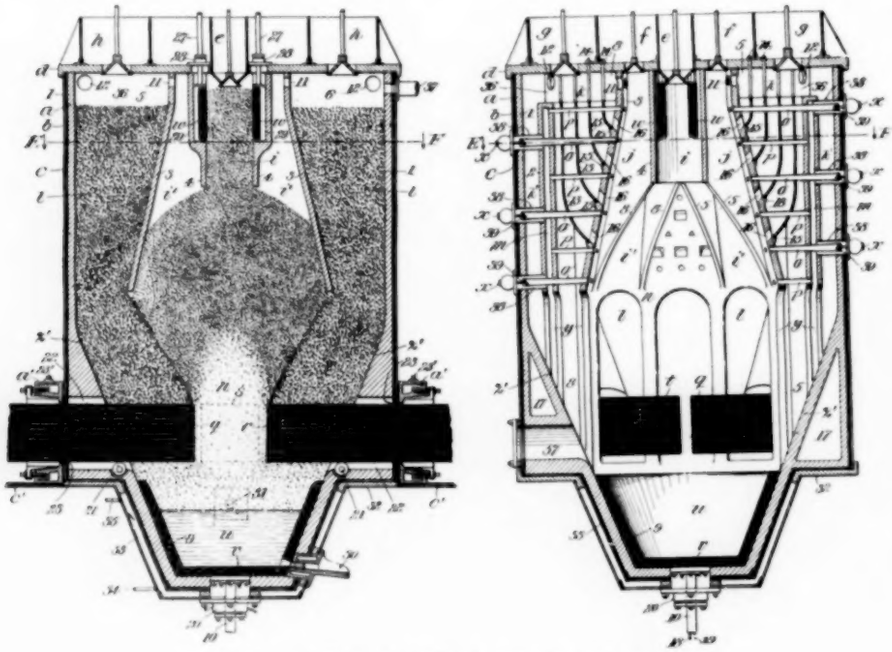
For the production of low-carbon metals the reducing agent should contain a minimum of titanium carbide since the oxidation of the combined carbon is not complete. In order to fully oxidize and utilize all of the titanium it is necessary to react on an excess of the reducible oxide. The resulting slags then contain both the oxidized titanium and the unreduced oxide of the charge. Titanium oxide not being acted upon by acids, the unreduced oxide may be dissolved out of the slags and recovered by evaporation. Tungstic oxide is readily dissolved out by hydrochloric acid. (992,422, May 16, 1911.)

Silicon Carbide, Silundum.—Mr. FRANK J. TONE, of the Carborundum Company, of Niagara Falls, N. Y., has formerly described a method in which shaped pieces of carbon are heated in proximity to a silicon-containing substance to such a temperature that the silicon combines with the carbon *in situ*, so that the shaped pieces of carbon are transformed into dense silicon carbide. (In Europe this is called silundum.) Mr. Tone has now found that if he makes his original shaped pieces not of carbon, but of a mixture of silicon carbide and carbon, he can vary the porosity and resistance of the product within wide limits. A mixture of ground silicon carbide and carbon, with tar, glue or silicate of soda as binding material, is shaped into the desired form and baked to remove moisture and other volatile materials. The article is then placed in an electric furnace and is embedded in a mixture of carbon and silica, or of silicon carbide and silica. The furnace is brought up to the temperature of the formation of carborundum, at which temperature vapors of silica and silicon are formed, which penetrate into the molded article to a considerable distance. These vapors reacting with the carbon change it into silicon carbide. The binding material is volatilized and the resultant product consists of silicon carbide bonded with silicon carbide in a dense, compact form. (992,698, May 16, 1911.)

Titanium Alloys.—For the production of ferrotitanium, Mr. A. J. ROSSI, of Niagara Falls, N. Y., proceeds as follows: A bath of molten iron is first provided in the electric furnace and this bath is covered with a layer of lime, CaO. Upon this is placed a layer of a mixture of titanic acid and carbon. The titanic acid is reduced to titanium, $TiO^2 + 2\,C = Ti + 2\,CO$; the titanium sinks downward and alloys with the iron. At the same time calcium carbide is formed, $CaO + 3\,C = CaC_2 + CO$. The calcium carbide, having reducing properties, promotes the reduction of the titanic acid and at the same time protects the molten alloy both from alterations by oxidation and volatilization. Toward the close of the process an additional blanket of carbon is added on the layer of calcium carbide in order to insure a reducing atmospheric atmosphere. Instead of titanic acid, as described above, titanate of lime, CaO,TiO_2, may be used. (986,505, March 14, 1911.)

Continuous Electric Shaft Furnace.—After many years of

successful experience with his electric furnace for carbon bisulphide manufacture Mr. Edward R. Taylor, of Penn Yan, N. Y., has undertaken in recent years to design an electric-shaft furnace which, while retaining the very satisfactory salient features of his carbon bisulphide furnace, would be especially applicable to reduction of iron ores in a continuous process. His latest design is shown in Figs. 3 and 4 in vertical section in two different planes, the second diagram showing the furnace empty. It is of stack form, the internal structure *a* being of brick or cement incased by a metallic shell *b* with an intermediate layer *c* of asbestos. *e, f, g, h* are feeding hoppers in the top, communicating with flue-shaped passages *i, j, k-k'* and *l* extending downward. The coke is introduced through the central hopper *e* into the passage *i*. The coarse ore is introduced through four hoppers *f* and passage *j*, symmetrically arranged around the center. The fine ore is introduced through the four external



FIGS. 3 AND 4.—ELECTRIC FURNACE FOR IRON ORE REDUCTION.

hoppers *g*, each of which communicates with two passages, a main passage *k* and a peripheral passage *k'*, separted from each other by hollow walls forming a gas space *m* within each wall, open at the bottom and also in direct communication with the working chamber *n* by way of outwardly flaring gas tubes *o*. The peripheral fine-ore passages *k'* communicate with the working chamber at their lower ends only. The main fine-ore passages *k* are provided with baffle plates *p* to retard the flow of the fine ore and permit gas spaces through the body of fine ore. Four other hoppers *h*, situated midway between the four external hoppers *g* and the peripheral passages *l*, are appropriated to the feed of resistance regulating material adapted to flow downward by gravity and to protect the main electrodes and regulate the conductivity of the charge at the heat zone. For this purpose broken electrode carbons may be used, crushed to suitable size.

There are four horizontal main electrodes *q, r, s, t*, arranged in two pairs at right angles to each other. They are located immediately above the hearth *u*. Secondary electrodes *v* and *w* are located at the bottom and top of the furnace respectively. An annular gas chamber 36 at the top of the furnace communicates freely with the coarse ore and coke passages and with the intermediate passages for fine ore and regulating material, and discharges through an outlet pipe 37. The tube *o* for circulating the CO and CO_2 gases within the furnace extend from the coarse-ore passages *j* outward into curbs 38, which project inward to the gas spaces *m* from outside the furnace, and electrically driven rotary blowers 39 are located within the outer ends of these tubes. To insure the inward feed of the ore stoking screws are provided, which are not shown in the illustrations here reproduced. Passages are also provided through which air may be blown into the center of the working chamber.

The height to which the central coke column is kept incandescent is determined by passing more or less current through it by way of the secondary electrodes *v* and *w*. This in turn determines the proportion of carbon in the molten metal as it reaches the hearth and facilitates producing either pig iron or steel at will. The gas-circulating blowers 39 insure the conversion of the carbon monoxide gas formed in the vicinity of the heat zone into carbon dioxide, the conversion of the latter into carbon monoxide, and so on. Considerable details are given in the specifications on the construction of the electrodes. (985,226, Feb. 28, 1911.)

Electrolytic Processes.

Electrolytically Refining Alloys.—Mr. Anson G. Betts, of Troy, N. Y., subjects an alloy which does not contain the two principal metals in a desired proportion to a separation into at least two other alloys, in one of which at least the ingredients are together in a more useful proportion. For this purpose the alloy is used as anode in an electrolytic cell, the two principal metals being dissolved, while many of the impurities are left as an anode slime. By varying the conditions of deposition he then alternately deposits at least two different alloys. Two examples are described, the refining of arsenic-antimony and of nickel-iron alloys.

The arsenic-antimony alloy is dissolved as anode in a solution containing antimony and arsenic trifluorides, an alkaline fluoride and hydrofluoric acid. When the solution is rapidly agitated arsenic deposits out faster than antimony and the cathode metal may thus contain more arsenic in proportion than is dissolved from the anode, until the solution becomes impoverished in arsenic. If now the solution is kept at rest and electrolysis continued an alloy is produced for a time, containing a less proportion of arsenic than is dissolved from the anode. When arsenic has sufficiently accumulated in the electrolyte, the first operation with rapid agitation is repeated. In practice there should be at least two refining tanks in use, through which the solution flows continuously from one to the other, and in one of which the solution is well agitated, while in the other it is nearly at rest. The alloys deposited in the two tanks are different from each other and from the anode and are generally also free of many impurities of the anode, as silver, lead, copper, etc. About one-half of the arsenic in the anode remains in the anode slime.

Iron-nickel alloys are treated as anodes in a solution of the sulphates, fluosilicates or other usable salts of iron and nickel. By varying the temperature the composition of the cathodic deposit is varied. Most of the impurities are eliminated as anode slime or remaining in solution. With an anode containing 6 per cent nickel, 6 per cent copper, 1 per cent manganese, 3 per cent carbon, 1 per cent silicon, by electrolyzing at a tem-

perature of 40° C. to 50° C. he deposits for a while cathode metal with 3 per cent nickel, while the percentage of nickel in solution increases. Then by electrolyzing at a temperature of 90° C. there deposits on the cathode an entirely different product, which may contain 13 per cent nickel and nearly 87 per cent iron, while the percentage of nickel in the solution diminishes. The solution is then cooled and the first operation repeated. In practice the two steps would be carried out in two different tanks.

"Nickel steels containing 3 per cent to 3.5 per cent of nickel are at present the most popular. One of the above-mentioned products of about this composition is directly convertible to an especially fine nickel steel nearly free from harmful impurities containing this amount of nickel, by melting with the proper additions, while the other product may be melted with other relatively impure iron to produce a lower grade of nickel steel. If the entire product made from anode metal of the composition given was of one kind containing about 7 per cent of

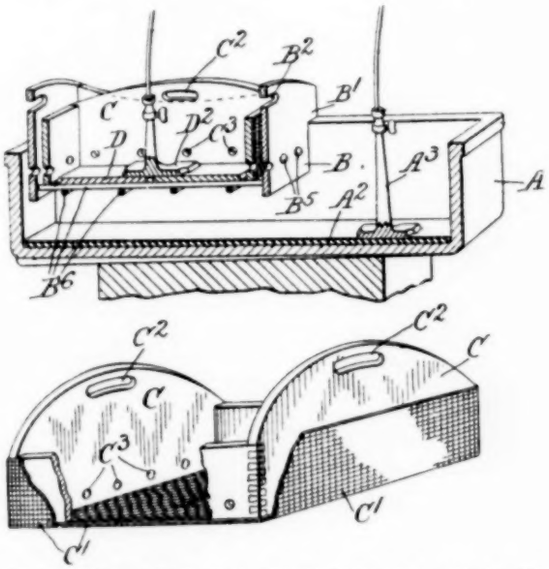


FIG. 5.—ELECTROLYTIC CELL FOR PARTING GOLD AND SILVER.

nickel it would require to be melted with outside non-electrolytic iron of lower grade to produce the very useful 3 per cent to 3.5 per cent nickel steel, and obviously would not be as good for making high-nickel steels as the 13 per cent alloy, for example." (987,947, March 28, 1911.)

Electrolysis of Brine Solutions.—In the electrolysis of sodium chloride solutions in diaphragm cells, like the Hargreaves cell, the solution coming from the cathode compartment contains undecomposed sodium chloride. To recover this and use it again is the object of a patent of Mr. GEORGE A. GABRIEL, of Brooklyn, N. Y. (assigned to the Bleach & Caustic Process Company). A second object is to renew and purify the anolyte and to remove any gases from it. These objects are obtained in a cyclic process in which evaporators, centrifugal driers, filters, etc., are employed. There are thirty-four claims. Claim 8 reads as follows: "In an electrolytic process, in combination with the step of electrolysis, the process of treating the discharged electrolyte remaining in the catholyte, which comprises removing the undecomposed electrolyte by concentrating the main body of the products of decomposition, converting the remaining products of decomposition contained therein into substances which purify the raw electrolyte, and then submitting the purified electrolyte to said step of electrolysis." Claim 24 reads as follows: "In an electrolytic process, in combination with a continuous main circulation of the electrolyte discharged from a cathode chamber, for purifying and supplying the electrolyte, a local or auxiliary circulation of the anolyte, including the steps of electrolysis and filtering." (959,730, May 31, 1910.)

Electrolytic Parting Cell.—What may be considered as an interesting further development of the well-known Balbach cell for parting gold and silver is patented by Mr. WILLIAM THUM, of Hammond, Ind., and shown in section in Fig. 5. *A* is a tank with an inclined bottom on which rests the cathode plate A^2 (for instance, of carbon), connection to the outside circuit being made through the cathode terminal A^3. The anode cell *B* is suported in the tank *A* by means of the projecting ledges B^1. Perforations B^5 are provided in all the walls. The bottom of the cell consists of a grid made up of the bars B^6. The anode basket *C*, which is located within the anode cell and rests upon the grid B^6, is provided with the canvas bottom C^1, as shown in the lower diagram, the handle holes C^2 and the perforations C^3 in its sides and ends. The anode plate *D* is situated within the basket *C* and rests upon the spacing bars, which rest in turn upon the canvas bottom of the basket. D^2 is the anode terminal. A modification of this construction is also described, in which the anode cell and basket are made in one.

If silver bullion containing gold is to be treated an electrolyte of silver nitrate with free nitric acid is used. The electrolyte is contained in the tank *A* at such a level as to be slightly above the perforations B^5 and C^3 in the anode cell and anode basket respectively. The silver dissolves from *D* as nitrate and the heavy silver nitrate solution sinks through the canvas and runs down over the full length of the surface of the inclined cathode plate A^2 on which the silver is deposited in shape of loose crystals. The electrolyte, the specific gravity of which has thus been reduced, rises now upward again and flows back across the upper part of the tank toward and into the anode cell through the perforations B^5 and C^3. The gold in the anode plate *D* is not dissolved and accumulates in the shape of mud or slime resting loosely on the perforated bottom of the basket.

The cell may be similarly used for refining bismuth. In this case an electrolyte of bismuth chloride and free hydrochloric acid is used. The bismuth is obtained in form of more or less coherent easily removable crystals, having the shape of trees, mushrooms and the like. (969,921, Sept. 13, 1910.)

SYNOPSIS.

GOLD AND SILVER.

Cyaniding Tailings.—Details of the treatment of a 500,000-ton tailing dump at the Oroya Brownhill, in Western Australia, are given in the March *Journal* of the Chamber of Mines of Western Australia, by Messrs. STEVENS and DEGENHARDT.. A vacuum filter plant was erected at the end of 1908 for the retreatment of a great mass of filter press tailing. The process consists of belt conveying to the plant, mixing with spent solution, agitation with cyanide, vacuum filtration and precipitation on zinc shavings.

The tailing treated consists of a mixture of 92 per cent. of an all-slimed product from bromo-cyanide treatment, and 8 per cent. of roasted concentrate. It is fine, only 2.5 per cent remaining on a 100-mesh screen. The average value is about $2 per ton, and by agitation with weak cyanide solution an extraction of $1 is obtained. Of the extracted value, one-half is present as dissolved gold in the 6 per cent. moisture contained in the dump; the $1 remaining in the residue when discharged is not soluble in stronger cyanide solution, and can only be extracted by regrinding and concentration, which would not be profitable. The tonnage treated is 730 tons in 24 hours.

A movable belt conveyer is placed along the side of the old dump, and the material is run on to it by men working with picks and shovels. As the dump is removed the belt conveyer is placed closer to it, each move giving about 10,000 tons. Taken over a period of two years the cost of the belt has been about 0.8 cent per ton.

The dump is easily moved, and three men working with picks on the dump, together with two men shoveling from the bot-

tom, can run about 250 tons per shift on to the belt. The material runs easily at 41.5 degrees, but in wet weather the rills must be a little steeper.

At the mill the material is mixed in a vortex mixer with spent solution from the extractor boxes. These mixers have cast-iron impellers on 3-in. shafts and are driven at 350 r.p.m. by quarter-turn belts. From the mixer the pulp flows to the agitation vats, which are 17 ft. 6 in. diameter and 9 ft. deep. These vats have bottoms of 18-gauge plain galvanized iron and sides of 20-gauge corrugated galvanized iron, and last about three years. The bottom is usually the first part to wear out. Each vat has four angle-iron arms driven at 12½ r.p.m.

The spent solution tests 0.04 per cent. KCN, and lime to the extent of 1.5 lb. per ton of tailing is added in the mixers, to bring the protective alkalinity up to about 0.002 per cent CaO. The pulp is agitated for eight hours, and the solution tests 0.02 per cent. KCN when it is drawn off to the vacuum filters. Average cyanide consumption is 0.47 lb. of sodium cyanide per ton. Specific gravity of the pulp is maintained at 1.61.

The filter equipment consists of two vats each 10 ft. wide, 21 ft. long and 7 ft. deep, the bottoms being divided into two 60-degree discharge hoppers. The filter vats are filled by an 8-in. centrifugal pump, the wearing shaft of which lasts about 4½ months and the runner 9 months. It is operated 38 per cent. of full time by a 45-hp. motor. Each filter vat contains 34 frames 5 ft. 3 in. deep by 9 ft. wide and set 6½ in. between centers of frames. The frames will carry an average of 22 tons of dry slime at a charge, making cakes 1⅞ in. thick. Of the pulp charged into the vat 43 per cent. is removed as solution and cake, and 57 per cent. is pumped back. The cycle of operations is as follows: Filling vat, 5 minutes; forming cake, 30; emptying excess pulp and putting in wash, 8; washing, 30; emptying excess wash, 4; discharging cake, 7; total, 84 minutes. A vacuum of 27 in. is maintained during filtration, and the cake is discharged with 24 per cent. moisture and soluble gold equal to 6 cents per ton of dry slime.

The pregnant solution is clarified in presses and precipitated on zinc shavings. The consumption of zinc is 0.13 lb. per ton. Cyanide is added in the gold solution vat as strong solution to assist precipitation, and the solution entering the boxes contains not over 0.05 per cent. KCN. About one ton solution is precipitated for one ton of tailing treated. The total cost of breaking and treatment is about $0.52 per ton (2s. 1.28d.).

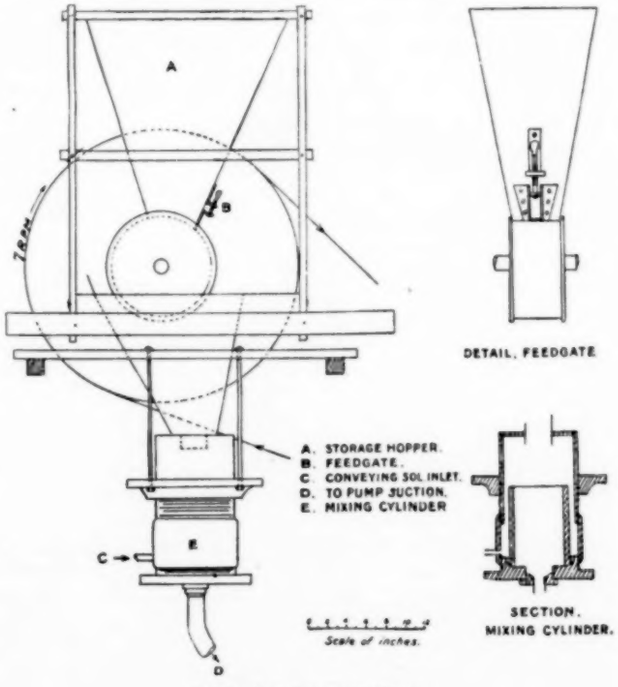
All-Sliming Treatment.—A fine example of the development of a cyanide process is given by Mr. Alex. Fyfe in a paper read before the Australasian Institute of Mining Engineers and published in the *Transactions* of that body. Beginning with stamp-amalgamation and separate treatment of slime and sand, with elimination of sulphides, which were treated separately, the process has been developed to a point where but one product is made and treated by cyanide. The change was gradual and resulted from numerous experiments, from each of which an excellent lesson was learned and applied.

The first step was an attempt to treat the sulphide concentrates at the mill instead of shipping them to smelters. The process included regrinding in tube mills, agitation with compressed air and filtering in presses. The cost of regrinding was high, as one tube 19 ft. 3 in. long by 4 ft. 8 in. in diameter could reduce only 10 tons in 24 hours, and the pebble consumption was 30 cwt. per week. Later coarse sand was added to the concentrate in an endeavor to improve grinding conditions, and this was so successful that the concentrate was fed into the mills grinding raw ore. Later amalgamation was dropped, and toward the end of 1908 cyanide solution was added to the mortars instead of clear water. This shortened the time of treatment and did away with the necessity of dewatering the slime before treatment with KCN, so that the vacuum filtration plant formerly used for this purpose was available for filtering pregnant solutions. Finally additional tubes were added and the sand ground to such a fine state that it was mixed with the slime and treated as one product. Thus from a complex system, with several products to treat, there has been evolved a simple process with but one product.

The solution added to the stamp mortars contains 0.10 KCN, and is added at the rate of 10 tons of solution per ton of ore. Lead acetate to the extent of 0.5 lb. per ton of ore also is added. The stamps weigh 1100 lb., drop 7 in. 105 times per minute, and have a duty of 7.6 tons per stamp per day. From the stamps the pulp flows to spitzkasten, the underflow from which goes to tube mills. The combined slime is settled and drawn off at a consistency of 1.5 solution to 1 of slime, and agitated in high tanks with air at a pressure of 38 lb. per sq. in. Time of agitation, 18 hours; cyanide consumption, 1.258 lb. per ton. Experiments showed that 87 per cent. of the total value is in solution before agitation begins, and that only a small additional extraction is effected in the tanks. The agitated pulp is then filtered in Moore filters and the solution precipitated in zinc boxes at the rate of 1.4 tons of solution per cubic foot of zinc in 24 hours. Expenses per ton of ore treated are distributed as follows.

Rock breaking	$0.104
Stamping	.166
Tube milling	.292
Classification (Wilfley tables)	.072
Treatment	.636
Precipitation and melting	.054
Assaying and sampling	.058
General expenses	.120
Total	$1.502

Zinc Dust Precipitation.—The formation of calcium carbonate during precipitation by zinc dust and the detrimental



ZINC DUST PRECIPITATION.

effects of this salt on precipitation are set forth by Mr. Allan J. Clark in the April, 1911, issue of the *Mining Magazine* (London). The formation of calcium carbonate may result from agitating, with compressed air, the solution before the precipitation of the precious metals, or by agitating with compressed air to form the zinc emulsion used for precipitation.

At the time the author's attention was first directed to the matter in hand, the method of preparing the zinc emulsion was that of agitating by compressed air in an inverted cone with a

continuous overflow to the suction of the pump, where it was mixed with the solution flowing to the presses. The first instance of the effect of calcium carbonate was noticed while precipitating a very weak and low grade solution of high alkalinity. This solution had been collected in shallow vats and agitated with compressed air, and was in prime condition for the formation of the objectionable salt. The precipitation offering some difficulty, the rate of flow was reduced without improvement in precipitation. The press was then opened, and a heavy deposit of calcium carbonate was found, caked on the press cloths or mixed with the precipitate. By eliminating the preliminary agitation of the solution the difficulty was remedied.

The author suggests that the formation of calcium carbonate diminishes the efficiency of the zinc by coating the zinc particles and preventing further action.

By eliminating the mixing cones and substituting the method of allowing the zinc dust to fall from an endless belt into a trough where it was washed by a stream of solution direct to the precipitation line, there was an increase in concentration of 63 per cent.; a decrease of consumption per ton of solution of 35 per cent.; and a decrease of zinc consumption per ounce of gold precipitated of 38 per cent.

Later, on account of the irregular action of the belt feeder, a new design of zinc dust feeder was installed. It is designed by W. H. Todd, foreman at the Homestake mill. It consists of a hopper with feed gate, and a mixing cylinder where a small stream of solution emulsifies the dust. No air agitation is used. Results of the first month's operation of the new feeder compared with the former method of emulsifying in a cone with air agitation show: Increase in concentration, 80 per cent.; decrease in consumption of zinc per ton of solution, 33 per cent.; decrease in consumption of zinc per ounce of gold precipitated, 31 per cent. The results warrant the statement that a reasonably regular supply of zinc is sufficient for all practical purposes when, as is usually the case, the zinc and solution travel some distance before entering the press.

Continuous Filter Press Process.

In the usual practice of handling oil seeds, fish scrap, slaughter house scrap, bagasse, etc., for the general purpose of removing the liquid from the solid, it is exceedingly desirable to employ a continuous process of operation, but in many types of apparatus now in use it is rather difficult to produce the necessary high pressure. With hydraulic or screw presses the pressure can be easily attained, but the process is essentially discontinuous. Any attempt to make the pressing process a continuous one is therefore of considerable interest.

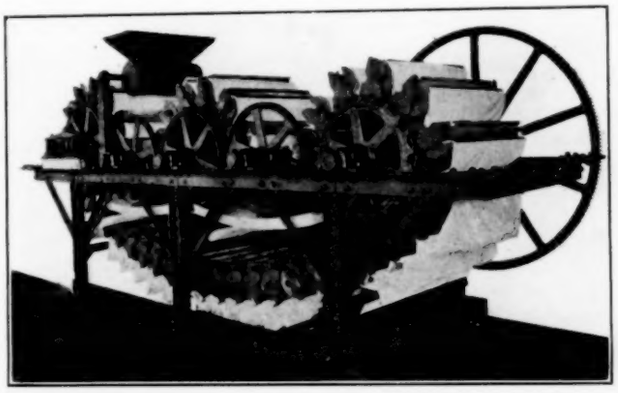
We had recently an opportunity to see in operation a new continuous pressing apparatus which is the invention of Mr. John J. Berrigan, of 136 North Centre Street, Orange, N. J. The apparatus which we saw in operation is reproduced in the adjoining illustration.

It will be seen there is an endless belt, running alternately around the top and the bottom of five successive rolls. But unlike other belt and roll arrangements, the closeness of the pinch between the faces of the belt and roll has nothing to do directly with the pressure to be exerted on the liquid and solid which are to be separated.

The belt is seen to carry a large number of pressure chambers. In the case of the apparatus illustrated their number is 54. The material to be separated is fed into them from the hopper shown on the top on the left of the illustration. Now each of the chambers is changed in shape while traveling with the endless belt around the rolls, that is the walls of the chamber are made to open or close like jaws while they are on top of a roll or on the bottom of a roll respectively.

When a certain chamber reaches the position on top of the first roll, it is wide open and is filled with material from the hopper. Then the filled chamber is carried by the belt downward and the walls are gradually closed, somewhat on the principle of a door jamb. When the chamber reaches the position below the second roll, the pressure becomes a maximum and the liquid is squeezed out and runs off through the filter cloth.

The chamber then begins to rise again and is again wide open when it reaches the top of the third roll. The pressure on the cakes is now relaxed, and at this point, according to requirements, sprays of water, steam or oil may be directed on the cake. When the chamber now begins to pass again downward, the cake in it is shifted from one side to the other side and thereby breaks up. The pressure begins to rise again, and when



CONTINUOUS FILTER PRESS.

the chamber reaches a position at the bottom of the fourth roll there is a second squeeze of liquid from the cake.

Then the chamber rises again and is wide open when it comes to the top of the fifth and last roll. But now after it has passed the top of the last roll and passes downward, it does not close, but on the contrary, when the belt with the chambers gets into the position below all the rolls, the filter cloth turns automatically inside out, as shown in the illustration, and the solid cake drops out.

The capacity of the machine depends on the number of chambers, the size of the chambers, and the speed. The speed is to be adjusted, according to the material under treatment. While the principle of the machine is thus clear, the reader may be referred for details of mechanical construction to three patents recently granted to Mr. Berrigan (994,631, 994,632 and 994,495, all of June 6).

The machine is certainly well designed mechanically. If pieces of wood or melting screws, etc., get incidentally into the charge no harm is done.

Perhaps most remarkable, however, is the small amount of power required since the machine shown in operation, weighing several tons, could easily be driven by hand. This is due to the well-balanced arrangement of the endless belt and accessories.

It will be decidedly worth while watching the introduction of this machine into commercial practice.

NOTES.

Messrs. **Toch Brothers,** the well-known manufacturers of paints, varnishes and enamels, of New York City, with works in Long Island City, have recently opened new works at Toronto, Canada, to supply the Canadian trade.

The **FitzGerald & Bennie Laboratories,** of Niagara Falls, N. Y., have been removed from Falls Street to the corner of Highland and Whirlpool Avenues, Niagara Falls. The new location affords greatly enlarged and improved facilities for the work of these laboratories.

The Elmore Concentration Plant at Sulitelma, Norway, using the oil-vacuum-flotation process, is being enlarged from

a daily capacity of 500 tons to about 900-1000 tons. This will be the largest flotation plant used for the treatment of copper ore. Common fuel oil is used, and the consumption is reported to be 2.7 lb. per ton of ore treated. No acid is used.

The Camp Bird, Ltd., reports that the January operations at its properties resulted in a net balance applicable to profit amounting to $96,515.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in December, 1910, was 68. The total tonnage milled during that period was 1,869,124 tons of ore; 9736 stamps were in operation with an average duty of 7.33 tons per 24 hours. Tube mills in commission numbered 190. The yield for the month was 640,995 fine ounces gold.

The Flow of Heat Through Furnace Walls is the title of Bulletin No. 8, recently issued by the Bureau of Mines. The results of the investigation described in this bulletin justify the following conclusions: In furnace construction a solid wall is better than a wall of the same total thickness containing an air space, as the former is a better heat insulator. This statement is particularly true if the space is close to the furnace side of the wall and if the furnace is operated at high temperatures. If it is desirable in furnace construction to build the wall in two parts, so as to prevent cracks being formed due to the expansion of the brickwork on the furnace side of the walls, it is better to fill the space between the walls with some "solid" (not firm, but loose) insulating material. Ash, crushed brick or sand offer high resistance to heat flow through walls with an air space. Furthermore, any such loose material by its plasticity reduces air leakage, which is an important feature deserving consideration.

The Magma Copper Company is one of the new copper propositions that has recently come into prominence. Unlike many of the Western copper mines, it is a high-grade proposition and will probably be a heavy producer when fully developed. It is in Pinal County, Arizona, 20 miles west of the Inspiration, 12 miles northeast of the Ray Consolidated, and is being developed at depth. The output of the mine will be largely smelting ore.

The Arizona Copper Company produced during the month of April, 1911, 1420 tons of copper.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment (*Continued*).

362,022, April 26, 1887, H. Liepmann, assignor to the Cassel Gold Extracting Company (Ltd.), of Glasgow, Scotland.

Similar to the preceding, except that the asbestos-cloth diaphragms are supplemented by movable diaphragms of earthenware or compressed layers of asbestos cloth to prevent diffusion of the alkaline liquors from the cathode back into the anode compartment. The supplemental diaphragms are movably carried by screws. In operation the powdered ores are placed in the revolving drum, or anode compartment, the dense partitions being forced up against the permeable ones. The drum being revolved and current supplied, the metals dissolve, but their solution is retained within the anode compartment. The supplemental diaphragms are then withdrawn, and lime added to the anode solution. The solution then passes through the permeable fixed diaphragms into the tanks, where the gold is electro deposited.

379,764, March 20, 1888, Charles F. Croselmire, of Newark, N. J.

Zinc. Pulverizes galena carrying zinc, silver and iron, roasts with an air of steam jet, and treats with dilute sulfuric acid. The zinc, iron and manganese dissolve. The latter are precipitated as oxids by injecting air into the solution. The zinc is then electrodeposited, or it may be precipitated as a carbonate or oxid. The undissolved residue is washed, briquetted and smelted.

387,036, July 31, 1888, Charles Peters Bellows, of Gloversville, N. Y.

Gold and silver. Treats sulfids, titanids, iridids, osmids, chromids, antimonids, to render them free-milling and capable of amalgamation. The ore, crushed to the size of fine gravel, is placed in a perforated crate or basket surrounded by cotton cloth. A copper anode bar is embedded in the ore. The basket is placed in an insulated tub of wood or porcelain containing a solution of sodium chlorid, to each gallon of which is added one ounce of caustic soda. Steam is led into the solution by an insulated pipe, and another steam pipe may extend into the gravel. Current is passed from the anode in the ore to a copper-sheet cathode surrounding the crate, thereby dissolving the refractory substances, or causing them "to chemically combine with the substances in the bath and remain in solution or be deposited in the ore as an innocuous salt."

391,360, Oct. 16, 1888, Henry H. Eames, of Baltimore, Md., assignor to John J. Kelly, of same place.

Gold and silver. Converts sulfied ores into chlorids by electrolytic chlorin, liberated from a solution of sodium or calcium chlorid, acidified by sulfuric acid or copper or iron sulfate. The cell is a circular iron vessel lined with Portland cement and constructed to resist internal pressure. Vertical electrode-plates of copper and zinc are supported on insulators at opposite sides of the cell. Centrally located in the cell is a circular casing for effecting circulation of the solution, having an inle at the bottom at one side and an outlet at the top at the other side. In one form, this casing contains a revolving agitator, and steam is supplied to the bottom of the vessel through an insulated pipe. In a modification, the casing contains a steam-coil connected to pipes extending through the top and bottom of the cell and having rubber insulating connections. The ore is pulverized to an impalpable powder and is placed in the cell outside of the casing, being kept in suspension by the circulation. In using the first cell, steam is supplied at a pressure of 25 pounds per square inch, maintaining a temperature of 250°.

BOOK REVIEWS.

Practice of Copper Smelting. By **E. D. Peters,** Professor of Metallurgy, Harvard University. Large 8vo., 703 pages, 121 illus. Price, $5. New York: McGraw-Hill Book Company

This is practically a revision to date of "Modern Copper Smelting"—a book so well known and appreciated that its merits need no eulogy. We can, therefore, confine our remarks to the main features of the revision.

Wet processes are outside the scope of the book; electrolytic refining is left out because of it being too large a subject, although it is an almost integral part of the smelting process of producing copper; foreign practice is almost altogether absent. We feel sure that if Dr. Peters were as familiar with foreign practice as he is with American he would have included more of it in his book—that we would have found Mansfield, for instance, represented by more than one little slag analysis. This is the principal shortcoming of the book; it informs Americans wonderfully well on what they are doing themselves, but gives very little information about foreign practice.

There is not a chapter in the book which is not full of valuable information; the work is a high-class treatise on modern American copper smelting practice, which no one interested in copper smelting can afford to be without.